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THE METALLURGY OF ZINC AND CADMIUM

BY

WALTER RENTON INGALLS

METALLURGICAL ENGINEER

AUTHOR OF PRODUCTION AND PROPERTIES OF ZINC

FIRST EDITION

The Engineering and Mining Journal

261 BROADWAY, NEW YORK

LONDON . . . . . 20 BUCKLERSBURY

1903

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THE ENGINEERING AND MINING JOURNAL

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TO

HIS WIFE

THIS BOOK IS DEDICATED BY

THE AUTHOR.

PREFACE.

Although the zinc industry of the world ranks in importance with the copper and lead industries, there has never been printed heretofore a special treatise on the metallurgy of zinc. Such essays on this subject as have been published in generally available form are embodied in the handbooks on general metallurgy and industrial chemistry, which being necessarily brief fail to describe adequately the modern practice in this branch of metallurgy. Besides these there is no literature on the subject save in the files of technical periodicals and the transactions of scientific societies, chiefly French and German, inasmuch as the Belgian and German engineers have led the way in this branch of metallurgy, until recently at least.

I have purposely given this treatise an encyclopædic character, going minutely into descriptions of many furnaces and processes which are old and out-of-date, and many new ones which have not been tried practically, partly because the metallurgy of zinc has not yet crystallized to the stage where certain methods can be definitely pronounced best or pointed out as indicating the only lines to follow in modern practice, and partly because a record of all that has been done is useful as a basis for new experimental work. There are many places where the roasting and distillation furnaces of the type of thirty years ago are still in use, not merely as the relics of an old practice, but as recent creations.

The metallurgy of zinc is peculiar in embodying many features of industries which do not belong to the domain of metallurgy alone. The calcination of calamine is a process which is analogous to lime-burning, both in its chemical and mechanical aspects; until recently the shaft furnaces used for both purposes were practically identical. The manufacture of the retorts required for the distillation of zinc ore is directly a branch of the pottery industry, and much is to be learned from a study of the latter. Furthermore there is a strong resemblance in some respects between the zinc industry and the glass industry, which is not surprising, inasmuch as the original Silesian distillation furnaces were built after the type of the glass furnaces of that day. Finally the zinc distillation furnace embodies the essential features of the retort furnace for the manufacture of illuminating gas;

the distillation furnace is in fact one in which a distillation of coal is carried out, but its chief object is the recovery of zinc and not the production of coal gas, and the latter, which is produced incidentally, is wasted in large quantities, no satisfactory method for its utilization having yet been devised.

Besides the analogies which the metallurgy of zinc presents with the other branches of industry enumerated above, it laps over into some others, so that no sharp dividing line can be drawn. Thus zinc ore occurs frequently in connection with lead ore and in the recovery of the two metals, the metallurgy of zinc is intertwined with the metallurgy of lead; especially is this the case in the treatment of the so-called mixed sulphide ores. The chief ore of zinc, blende, is also valuable as a source of sulphuric acid, and many works recover the latter as a by-product. The zinc metallurgist is consequently interested in the subject of sulphuric acid manufacture. Sulphuric acid manufacture is in itself, however, a subject which in technical magnitude is equal, if not superior, to the metallurgy of zinc, and in a treatise on the latter it would be obviously inadvisable to go extensively into the manufacture of the former; nor would such a digression be anything but futile, inasmuch as the manufacture of sulphuric acid has already been treated in a masterful manner by experts in that line. On the other hand, a treatise on the metallurgy of zinc which did not consider the question of making sulphuric acid from the gases derived from blende roasting would be seriously deficient. I have therefore summarized some of the data most useful to the zinc metallurgist, without making too great a digression from the main subject.

The subject of fuel and its combustion has been treated at great length because of its surpassing importance. The principles of the combustion of coal are important in many branches of metallurgy, but in zinc smelting they are most important. There is no other of the chief metals of commerce which is produced at the expense of so much coal per pound of metal as is the case with zinc. The consumption of coal in smelting zinc ore is so great that it is an axiom in the industry that the ore must be carried to the coal rather than the coal to the ore. Even under that condition the cost of the coal constitutes with labor the chief items in the expense of smelting; in one place the cost of coal is the higher, in another place the cost of labor. The zinc metallurgist is obliged therefore to direct his attention pointedly to economy in the consumption of coal.

In concluding this preface I have pleasure in acknowledging the valuable assistance that has been rendered me by many friends. Special acknowledgment is due to Professor H. O. Hofman and F. J. Falding, Esq., who read the proofs of certain chapters and made many valuable suggestions;

also to Signor Erminio Ferraris, of the Societa da Monteponi, who contributed the notes on the calcination of calamine in Sardinia, and M. Gaston St. Paul de Sinçay, of the Société Anonyme de la Vieille Montagne, for drawings of furnaces and other valuable information; and to Mr. Otto Rissmann, of the Cherokee-Lanyon Spelter Co., Mr. E. C. Moxham, of the Bertha Mineral Co., and Mr. E. C. Hegeler, of Lasalle, Ill., besides many others who have afforded me access to their works and information as to their processes. Indispensable assistance has been freely given by many manufacturers of machinery, who have courteously furnished drawings and contributed much valuable data.

Lynn, Mass., March, 1903. WALTER RENTON INGALLS.

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METALLURGY OF ZINC.

I.

ZINC AND ITS ORES.

Zinc as a metal was probably known to the ancients, but it was never in common use by them and such knowledge of it as was possessed was for a long time lost sight of. In the middle ages brass was made by the cementation process in important quantity, and mines of calamine were worked for that purpose, but no one appears to have been aware that a distinct metal existed in that ore. The word zinc was first used by Basil Valentine, but he did not refer to it especially as a metal. That it was known as such was first mentioned by Paracelsus (1493-1541). In 1550 Agricola recognized a metal accidentally produced in the furnaces at Goslar, in the Lower Harz, which he called zink, or conterfey, but he did not know that it came from calamine. In the sixteenth century zinc was brought to Europe from China and the East Indies under the name of tutanego and a specimen from that source was examined by Libavius (1595), who was the first to investigate the properties of the metal with any exactness, but the fact that calamine was an ore of zinc was not known until the next century.

In 1721 Henckel published his discovery that zinc could be obtained from calamine by distillation and he is considered to be the first who intentionally carried out the process in Europe. The production of zinc on an industrial scale was first undertaken, however, by John Champion, who erected works at Bristol, England, in 1740, or about that time. The process consisted in the distillation of the ore in large pots, a pipe extending downward from the bottom, and this became known later as the English process, which continued in use until about 1860. It is said that the process was of Chinese origin, having been introduced at Bristol by Doctor Isaac Lawson, who went to China expressly to study it. Toward the end of the century Johann Ruhberg, of Pless, in Silesia, learned the art of zinc smelting in England, and in 1799 or 1800 he constructed a zinc furnace at Wessola, utilizing for this purpose the pots of a wood-fired glass furnace. This was the beginning

of the Silesian zinc smelting industry. In 1799 Bergrath Dillinger erected a smeltery at Döllach, in Carinthia, employing small vertical pipes for the distillation, a method (known as the Carinthian) which did not long survive. In 1805 the Abbé Daniel Dony, a chemist of Liège, Belgium, discovered independently the method of zinc smelting and from his experiments the Belgian zinc industry developed. Zinc smelting was not attempted in the United States until 1850.

PROPERTIES OF ZINC.—Zinc is a bluish white metal of shining luster, which in the periodic arrangement of the elements by Mendeléef is classified in the group with beryllium, magnesium, calcium, strontium, cadmium, barium and mercury. In its chemical relation, it is bivalent, combining with two atoms of chlorine and one atom of oxygen, forming a chloride and an oxide which are analogous to those of magnesium and cadmium. The atomic weight of zinc is 65·4; the atomic volume, 9·1.

Physical.—The specific gravity of cast zinc when poured at or near its melting point and cooled slowly is about 7·125. It is a trifle higher when cooled quickly and a trifle lower when poured at red heat. By rolling, the specific gravity is increased to 7·2, a cubic foot of rolled zinc weighing about 450 lb. According to the latest investigations, the melting point of zinc is 415° C., and the boiling point 920° C. The boiling point is raised 1·5° C. for each centimeter of atmospheric pressure above 760 mm. The fluid density of zinc is 6·48 to 6·55; the specific gravity of zinc vapor is 2·36. In melting zinc, the increase in volume from the cold, solid state to the liquid is 11·10%. Between 0° and 100° C. the specific heat of zinc, according to Regnault, is 0·09555; other authorities give it as 0·0927 from 0° to 100° C., and 0·1015 from 100° to 300° C. Metallic zinc burns in the air at a temperature as low as 500° C.

Zinc crystallizes in rhombohedral forms of the hexagonal system; also according to the isometric system. Ordinarily the metal has a coarsely laminar texture, sometimes granular. The fracture is granular or coarsely crystalline, dependent upon the temperature of casting, being coarse when the metal is heated nearly to redness before pouring and fine when the temperature of the liquid metal is not much above its melting point. At ordinary temperatures zinc is brittle, especially when impure, but between 100° and 150° C. it becomes so malleable that it may be rolled into sheets, and so ductile that it may be drawn into wire; after cooling it retains those properties. In malleability zinc ranks between lead and iron; in ductility between copper and tin. It is difficult to file, and when bent after fusion emits a crackling sound like the cry of tin, but not so loud. Its hardness is 2·5 according to Moh's scale.

The thermal conductivity of zinc is 28·1 (silver being 100); its electrical conductivity is 16·92 (mercury at 0° C. being unity). The co-efficient of linear expansion is 0·0000291. Its tensile strength is 7000 to 8000 lb. per square inch (Roberts-Austen).

Chemical.—In dry air zinc retains its luster, but in damp it becomes covered with a thin grayish-white coat of a basic carbonate, which protects it from further oxidation. Zinc is electro-positive to all other common metals, except magnesium, and precipitates all the ductile metals from their solutions, except magnesium, iron and nickel. Impure zinc is readily dissolved by nitric, chlorhydric, sulphuric and sulphurous acids; also by the caustic alkalies. Pure zinc is almost unaffected by acids, except nitric. At high temperatures zinc unites with oxygen, chlorine, bromine, iodine and sulphur. It is oxidized by carbonic acid, by lead oxide, and by the carbonates of the alkali metals.

Impurities Occurring in Zinc.—The common impurities of commercial zinc are lead, iron and cadmium. Small proportions of arsenic, antimony, carbon and sulphur also occur. Copper, tin and silver are noted in some spelters, and rarely minute quantities of thallium, indium, gallium, magnesium and aluminum have been observed.

Lead is found in most makes of spelter, often to the extent of 2 or 3%, but by refining the tenor can be reduced to 1 or 1·5%. A moderate percentage of lead increases the ductility and malleability of zinc, but with an increasing proportion the metal becomes tender. A tenor of 1·5% Pb will permit zinc to be rolled without cracking the sheets, but will unfit them for many purposes. Lead is also objectionable in spelter to be used for the better grades of brass, but a moderate percentage does not unfit it for use in making the common and inferior grades.

Iron tends to make zinc less fluid, less malleable, less strong and harder and more brittle, its bad effects becoming strongly marked when the percentage is as high as 0·20. Commonly the percentage of iron in commercial zinc is 0·01 to 0·05. The brass trade imposes a limit of 0·05%.

Cadmium seldom occurs in spelter in more than minute quantity and has no deleterious effect, except in the case of spelter which is to be used for making zinc white; in the latter case the product is likely to be discolored by cadmium oxide.

Copper and tin make zinc harder and more brittle. Arsenic makes it brittle and difficult to melt. Sulphur and carbon do not appear to have any injurious effect, at least not in the small proportions that commonly occur. The black flocks which remain after dissolving commercial zinc in sulphuric acid consist chiefly of lead sulphate and carbon.

ZINC ALLOYS.—Zinc alloys with aluminum, antimony, bismuth, copper, gold, iron, lead, magnesium, mercury, nickel, silver, sodium, tin and some of the rare metals. Of the binary alloys, only the aluminum-zinc and copper-zinc series have great industrial importance. The former are recent inventions; the latter constitute the various brasses and yellow metals.

The alloys consisting of one third zinc to two thirds aluminum and one fourth zinc to three fourths aluminum are remarkable for their strength and lightness, and are the more valuable because of the clean and sharp castings which can be made and the facility with which the metal can be machined.

Zinc and copper unite in all proportions, but only the alloys with upward of 50% Cu. are of industrial importance. The ductility and malleability of these alloys increase with the percentage of copper, and the best kinds, such as red brass, gilding metal, and percussion cap metal are made to contain 80 to 96% Cu. Dutch metal contains 80 to 85%; Prince's metal, 75%; best English sheet brass, 70 to 72%; ordinary brass, 66·67%, and Muntz's or yellow metal, 60%. The last, although somewhat deficient in ductility and toughness, possesses the great advantage that it can be rolled either hot or cold.

CHEMICAL COMPOUNDS OF ZINC.—The chemical combinations of zinc which play an important part in the ordinary method of recovering the metal are the sulphide, the sulphates, the oxide, the carbonates, and the silicates. Of less importance are the sulphites, ferrate and aluminate.

Sulphide—Zinc sulphide is infusible and at moderate temperatures non-volatile, but according to Percy it volatilizes at high temperatures. It can be roasted in an atmosphere of air or steam, giving off sulphurous anhydride and hydrogen sulphide respectively, but the reaction in steam is incomplete and in air there is a tendency toward the formation of sulphates, which can be broken up, however, by a sufficient increase of temperature. When strongly heated with iron or lime, zinc sulphide is to some extent decomposed, zinc vapor being liberated.

Zinc sulphide is decomposed and dissolved by mineral acids, nitric being the best solvent. Concentrated sulphuric acid oxidizes it to sulphate, sulphurous anhydride and sulphur being formed in the reaction. Chlorine vapor or water attacks zinc sulphide feebly, but at moderately high temperature zinc chloride and sulphur monochloride are formed, and at still higher temperature (600° C.) zinc chloride and free sulphur.

Sulphates.—There is a neutral sulphate of zinc ($ZnSO_4$) and several basic sulphates, corresponding to the symbol $ZnSO_4$, n ZnO. Both kinds exist as hydrous and anhydrous salts, the common form of the hydrous, neutral salt being $ZnSO_4+7H_2O$, which loses six molecules of its water at 100° C. and

the last one at 200° to 250°. The neutral sulphate is very soluble in water; the basic sulphates are insoluble, or but slightly soluble. Neutral zinc sulphate is decomposed by heat into oxide and sulphuric and sulphurous anhydrides; generally some basic sulphate is formed in the process and further decomposed by continued heating. At dull red heat zinc sulphate is reduced by carbon to oxide; at white heat, to sulphide.

Sulphites.—Zinc oxide unites with sulphurous acid to form monosulphite, which is insoluble in water, but is dissolved as bisulphite by an excess of sulphurous acid. Sulphurous fumes from roasting furnaces have been neutralized in this manner. When a solution of zinc bisulphite is boiled, sulphurous anhydride is expelled and the monosulphite is precipitated. This salt is converted to oxide by a gentle calcination. On exposure to the atmosphere it changes to sulphate.

Oxide.—Zinc oxide is infusible and is non-volatile at moderate temperatures, but according to Stahlschmidt it is notably volatile at 970° C., about 15% more so at 1054° C., and rapidly at white heat. It is reduced by carbon, carbon monoxide and hydrogen; the reduction by carbon begins at about 1100° C. When zinc oxide is reduced by hydrogen, steam is formed, which may under certain conditions reoxidize the zinc vapor. Metallic iron is said by Percy to reduce ZnO at high temperatures.

Zinc oxide is soluble in all the common mineral acids, and in many of the organic. It is also soluble in the caustic alkalies and in ammonium carbonate. Ferric chloride and sulphate dissolve it as chloride and sulphate respectively, ferric hydroxide being precipitated. Zinc oxide suspended in water is attacked by chlorine gas passed into the emulsion, the zinc going into solution as chloride and hypochlorite.

Hydroxide.—This compound is precipitated from solutions of zinc by means of the caustic alkalies, an excess of those reagents being avoided. It is also thrown down by milk of lime and milk of magnesia. The hydroxide is easily converted to oxide by calcination.

Carbonates.—Zinc forms numerous carbonates, neutral and basic, hydrous and anhydrous. The neutral carbonate, $ZnCO_3$, occurs in nature as the mineral smithsonite, but it is not easily prepared artificially. When zinc is precipitated by means of sodium or potassium carbonate, the product is commonly a basic carbonate. The carbonates of zinc are readily soluble in acids and are easily dissociated by heating.

Silicates.—Zinc oxide combines with silica in various proportions, forming a singulo-silicate, bisilicate, etc. They are difficultly fusible, the more so the higher their tenor in silica. According to Percy, the bisilicate is infusible at the most intense white heat. Zinc silicate is reduced by carbon.

Aluminate.—Zinc aluminate (ZnO,Al_2O_3) is formed when alumina and zinc oxide are heated together at high temperature. It is frequently formed in the walls of the retorts for distillation, which are thereby colored blue.

Ferrate.—Zinc ferrate (ZnO,Fe_2O_3) is formed when zinc oxide is heated with ferric oxide under certain conditions. There is reason to believe that it is produced to some extent in roasting ferruginous blende.

Chlorides.—The neutral chloride of zinc, $ZnCl_2$, is a white, deliquescent substance, which is very soluble in water. It melts at 262° C. and boils at about 710° C., vaporizing without decomposition. It is, however, difficult to boil down an aqueous solution of zinc chloride without setting free chlorhydric acid and producing some basic chloride. Basic chlorides are also formed to some extent when a solution of zinc chloride is precipitated with milk of lime or milk of magnesia.

ORES OF ZINC.—The principal ores of zinc are blende and calamine, the latter including the carbonates—smithsonite and hydrozinkite—and the silicates—willemite and hemimorphite. In the United States the New Jersey deposits of franklinite (mangano-ferrate of zinc) and zinkite (oxide of zinc) are important, but those minerals do not occur elsewhere in workable deposits. Data of the chief zinc minerals are summarized as follows:

Mineral	Hardness	Spec. Grav.	Symbol	Zinc tenor
Blende	3·5 to 4·0	3·9 to 4·1	ZnS	67·15%
Franklinite	5·5 to 6·5	5·1 to 5·2	$[(Fe,Zn,Mn)O,(Fe,Mn)_2O_3]$	21·00%
Hemimorphite	4·5 to 5·0	3·4 to 3·5	$2ZnO,SiO_2+H_2O$	53·70%
Hydrozinkite	2·0 to 2·5	3·6 to 3·8	$3ZnO,CO_2+2H_2O$	57·10%
Smithsonite	5·0	4·3 to 4·5	$ZnCO_3$	52·00%
Willemite	5·5	3·9 to 4·2	$2ZnO,SiO_2$	58·10%
Zinkite	4·0 to 4·5	5·4 to 5·7	ZnO	80·25%

These minerals seldom occur pure. Besides the silicious, calcareous and earthy gangue with which they are associated, and the other metallic minerals, like galena, pyrite, marcasite and limonite, the zinc minerals themselves are generally contaminated by chemically combined impurities, especially iron. Thus, almost all blendes contain iron as monosulphide, the proportion being upward of 10% in the variety called marmatite. Many blendes are cadmium-bearing, although the proportion of that element rarely exceeds 1%. The zinc minerals can be separated from the impurities with which they are mechanically associated by a suitable method of dressing to a more or less extent, and such a concentration is almost always a preliminary to the smelting process. The chemically combined impurities cannot, however, be separated by any mechanical process and therefore go with the mineral into the smelting process.

II.

CALCINATION OF CALAMINE.

Calamine, whether it be carbonate or silicate of zinc, should be calcined in order to prepare it for reduction and distillation, for the purpose of driving off the carbonic acid and water in the case of the carbonate and the water in the case of the silicate, both carbonic acid and aqueous vapor being undesirable in the retorts on account of their oxidizing action on the zinc vapor. Raw smithsonite might be charged directly into the retorts and zinc could be got from it, but aside from the disadvantages above mentioned the expulsion of the carbonic acid in the retorts would be effected in an objectionable manner, causing loss of heat and otherwise interfering with the process, wherefore it is best to subject the ore to a preliminary calcination in a special furnace. Zinc silicate can be reduced directly by carbon, and in the case of willemite, the anhydrous silicate, no preliminary treatment at all is required. The hydrous silicate on the other hand should be burned sufficiently to eliminate its water of constitution, although even that is not absolutely necessary. Thus the silicate ore of Missouri, which used to be calcined, is now charged quite raw into the retorts without there being any apparent difference in the metallurgical results. It is doubtful, however, if this conclusion would be supported by careful metallurgical tests.

The calcination of calamine, having been given up in the West, while the New Jersey ore being anhydrous does not require it, is comparatively unimportant in the metallurgy of zinc in the United States. In Europe, on the other hand, calamine ores are generally calcined before smelting, and inasmuch as they continue to be abundant, although their proportion in the total supply of ore is diminishing every year, the process of calcination is a matter of some interest. It is performed frequently at the mines before shipment of the ore in order to reduce the weight and save freight.

Principle and Practice.—The calcination of zinc carbonate is in principle and practice precisely analogous to the burning of limestone to lime, i.e.,

the ore must be heated to such a temperature and for such a length of time as is necessary to drive off the combined carbonic acid. With zinc carbonate this can be accomplished at a lower temperature and with less expenditure of heat than is required in the case of calcium carbonate. The temperature of dissociation of various metallic carbonates is given in the following table:

Substances.	Temperature.c
$ZnCO_3 = ZnO + CO_2$	a 300° C.
$MgCO_3 = MgO + CO_2$	b 650° C.
$FeCO_3 = FeO + CO_2$	b 800° C.
$CaCO_3 = CaO + CO_2$	b 812° C.

a Watt Dictionary of Chemistry, who states that, according to Rose, $ZnCO_3$ slowly evolves CO_2 at 300° C
b Le Chatelier, Thon. Ind. Ztg. 1886, p. 429. c Temperature at atmospheric pressure.

Practically the calcination of zinc carbonate is effected at a moderately red heat, while for the burning of limestone a bright red is required.

If the calcination be complete, pure smithsonite loses 35·2% in weight; while hydrozinkite loses 24·7%, and hemimorphite only 7·5%. The reduction in weight which can be effected practically depends, of course, upon the efficiency of the calcination and the character of the ore. If the ore has a quartzose gangue that portion of it will suffer no loss and the reduction in weight will be lessened according to the proportion in which it exists. On the other hand if limonite, or other hydrous minerals, be present they will lose their combined water, while calcite and dolomite may lose some of their carbonic dioxide if the calcination be performed at a high temperature. In no case, however, is the elimination of the carbonic dioxide likely to be complete, not even with a high grade zinc carbonate, which is the most easily dissociated carbonate, and the percentage of carbonic dioxide remaining after calcination may be as high as 15.

Besides the reduction in weight which is effected by elimination of the carbonic dioxide and chemically combined water, a saving in weight is made also by the expulsion of the hygroscopic water incidentally in the process of calcination. Ore as delivered from the mine does not have to be very wet to show 10% of hygroscopic moisture and ore which is apparently almost dry is likely to show as much as 5%.

The desirability of calcining calamine ore before shipment from the mine increases with the distance of the latter from market and the magnitude of the freight charges in which economy can be effected. If an ore contains 10% hygroscopic moisture and after drying the weight can be still further reduced 25% by calcination, a ton of crude ore can be brought

down to 2000×0·90×0·75=1350 lb., a saving of 650 lb.=0·325 ton. If the freight rate were $5 per ton and the cost of calcination $0·50 there would be a saving of (0·325×$5) −$0·50=$1·125 per ton of crude ore, besides which the cost of calcination would be partially, if not wholly, offset by a lower smelting charge.

After burning a carbonate ore the zinc oxide resulting has the property of absorbing carbonic acid from the atmosphere, although much less rapidly than is the case with lime. Owing to this tendency on the part of lime, calcareous ores especially should not be held for a long time before turning them over to the reduction furnaces.

In the calcination of zinc silicate ores it is only necessary to heat them sufficiently to drive off the hygroscopic and combined water.

The calcination of either variety of calamine may be effected in shaft furnaces or reverberatory furnaces, the former being especially applicable to coarse ores and the latter to fines. Any variety of fuel—wood, coal, petroleum or gas—may be used with either, inasmuch as heat only is required, the fuel having no other function than to produce the requisite temperature. In Upper Silesia, the waste heat of the distillation furnaces is still utilized for the calcination of calamine at some works, but the practice is no longer so common as it used to be, partly because of the more general adoption of Siemens furnaces and partly because of other considerations.

Analogy between Calamine Burning and Lime Burning.—On account of the analogy between calamine burning and lime burning and the fact that many zinc ores contain considerable quantities of calcite or dolomite, of which the carbonic dioxide has to be eliminated, it is advisable to consider in this connection the principles of lime burning. Calcium carbonate when heated gives up its carbonic dioxide only in the presence of air; without air, even when heated intensely, it simply fuses, its chemical composition remaining unchanged; with proper supply of air the normal temperature for successful burning is about 850° to 900° C., and the operation seems to be facilitated by the presence of aqueous vapor, at least in burning limestone in lumps as is generally the case, for which it is sufficient to keep water in the ash pit of the kiln, if it be one of the grate-fired type; otherwise the injection of sprays of water or steam into the kiln is recommended. It is stated that the use of steam has been found to be similarly advantageous in the calcination of zinc carbonate.

Calcination in Shaft Furnaces.—The calcination of calamine in shaft furnaces has the advantages of proportionately small consumption of fuel, ability to handle lump or run of mine ore, and large output with compara-

tively little labor. The proportion of fines which can be calcined ranges up to 20% of the weight of the lump ore, depending upon the skill of the workmen in arranging them so as not to interfere with the draught. The types of kilns employed are the same as are used in lime burning, but gas and petroleum firings, which are now employed extensively in lime burning, do not appear to have yet been applied in the calcination of calamine.

Types of Furnaces.—Shaft furnaces for the calcination of calamine are designed with independent fireplaces; or without fireplaces, the ore and fuel being then charged in alternate layers. The former may be fired with wood, coal, petroleum or gas; the latter may have grates or no grates, the

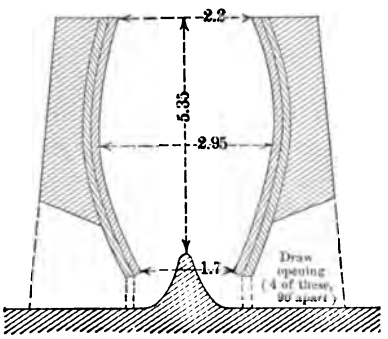


FIG. 1. OLD FORM OF SHAFT FURNACE WITHOUT GRATE, USED AT MORESNET.

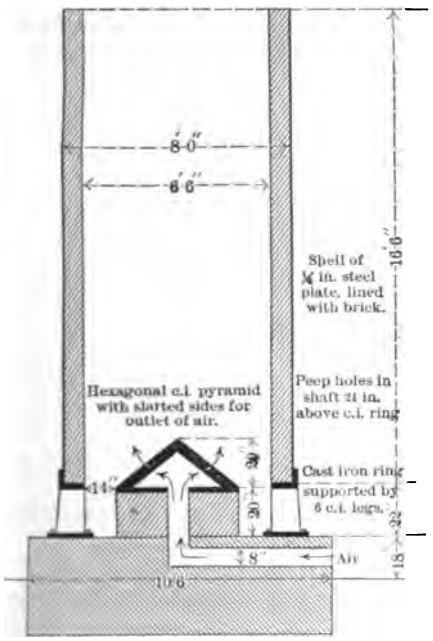
The dimensions are in meters. The furnace had a capacity of 25,000 kg. of crude ore per 24 hours, which lost 27% in weight and was drawn at intervals of four hours. The coal consumption was 3 to 4% of the weight of the crude ore.

column of ore resting directly on the hearth. Furnaces of the various types are illustrated in the accompanying engravings, which are self-explanatory.

The simple shaft kilns used in Europe vary from 10 to 26·5 ft. (3 to 8 m.) in height and 4 ft. 4 in. to 10 ft. (1·3 to 3 m.) in diameter. The oldest kilns are contracted at both top and bottom, the shaft having the shape more or less of a barrel. The more modern kilns have a simple cylindrical shaft or a conical form, the larger diameter being at the bottom and the walls sloping inward about 3% from the vertical. In externally fired kilns the shaft is usually boshed in toward the bottom.

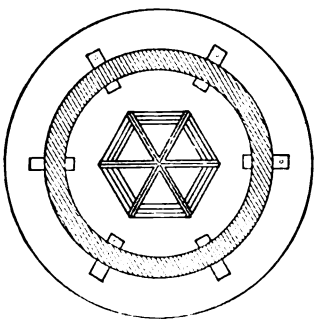
Advantages and Disadvantages of Internally Fired Kilns.—The design of the kiln and manipulation of the charges are the chief factors in the cost of burning calamine, since they determine the quantity of fuel re-

quired, the latter depending upon whether the carbon is converted by combustion into monoxide or dioxide (the more the proportion of dioxide the greater the development of heat) and the extent to which the heat so developed is utilized. A large amount of heat may be wasted by imperfect draught or bad firing. Kilns without fireplaces, in which the fuel is charged in layers alternating with layers of ore, are the cheapest to construct and are the more economical of fuel, but they lead to a slight deterioration of the grade of the ore through commingling of the ash, and may lead to loss of zinc by reduction and volatilization if the temperature is



FIGS. 2 AND 3.
SHAFT FURNACE FORMERLY USED IN KANSAS FOR THE CALCINATION OF CALAMINE.

Fig. 2. Vertical section. Fig. 3. Plan.



carelessly permitted to rise too high. Kilns heated from external fireplaces are free from the above disadvantages, but they require a larger proportion of fuel.

Method of Charging.—In charging a kiln of either type it is best to have the ore in pieces of nearly uniform size, if that can be managed without too much difficulty, since the calcination of the large pieces requires a longer time than the small, and if the latter are mixed in they have to remain in the kiln for some time after they are burnt, occupying useful space and consequently reducing the output. If the ore can be divided cheaply into several sizes it saves time and fuel to burn each separately.

A certain proportion of fine ore may be mixed in with each charge to fill the interstices between the lumps, but just as in charging a pyrites kiln great care is needed in the arrangement of such material in order not to check the draught. The more part of it should be placed around the periphery of the shaft where the draught is strongest.

When the kiln is to be charged with ore and coal alternately the ore should be put in layers of 6 to 8 in. with the proportional weight of coal (3 to 10%) intervening. In order to insure complete combustion care should be taken to distribute the fuel uniformly and the individual charges should not be too large. Too large and unevenly distributed charges are likely moreover to lead to imperfect burning of the ore. The drawing of the ore at the proper time is of great importance in such a kiln, since if it

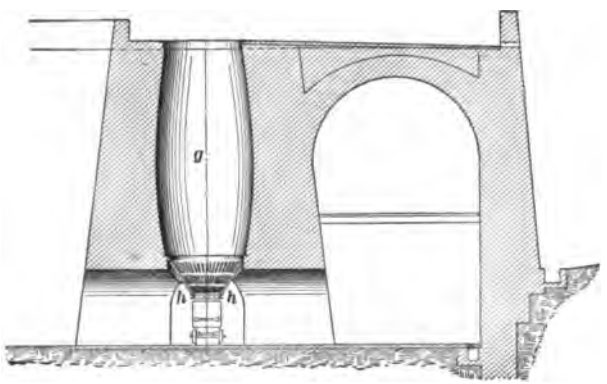


FIG. 4. OLD FORM OF SHAFT FURNACE WITH GRATE, USED IN SARDINIA.
The furnace shown in this engraving was 8 m. in height from the ground to the charge floor.

be drawn too slowly the fire rises too far and the kiln cools off in its lower zone, while if it be drawn too rapidly the ore is imperfectly burned. Regularity in charging and discharging is of great importance in the successful operation of a kiln.

To start the kiln it is filled with ore to about two thirds of its height, when a layer of shavings and wood is introduced (about 3 ft.) and ignited; coal is then added and upon this a thin layer of ore is placed. When the coal is burning well a larger charge is put on above the ore, and the alternate charging with coal and ore is continued until the kiln is full. As soon as flame shows near the top the drawing is commenced, and the amount of coal per charge is reduced to the regulation quantity (vide also p. 20).

To interrupt the burning process, once it has been started, the draw

openings are sealed and the top of the kiln is covered with fines. In doing this the draw openings should first be closed, before covering the top.

Design of Externally Fired Kilns.—In externally fired kilns no fuel is interposed with the ore and the heat required for the calcination is obtained from one or more outside fireplaces. A single fireplace is suitable only for small kilns, of which the shaft is contracted narrowly at the bottom as in Fig. 5. With larger kilns, especially when the area at the fire line is comparatively large, two or three fireplaces, disposed equidistantly around

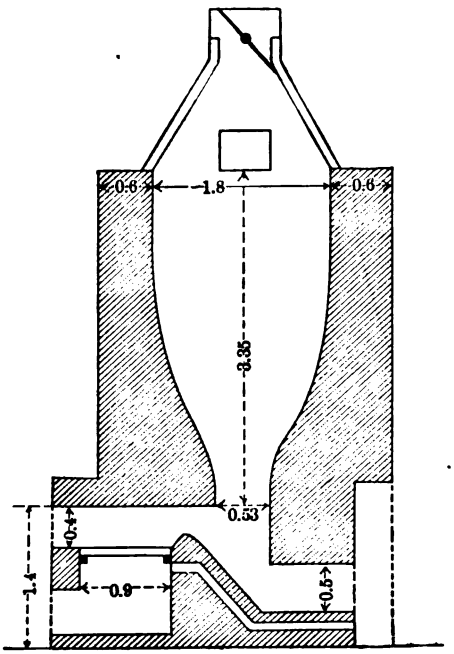


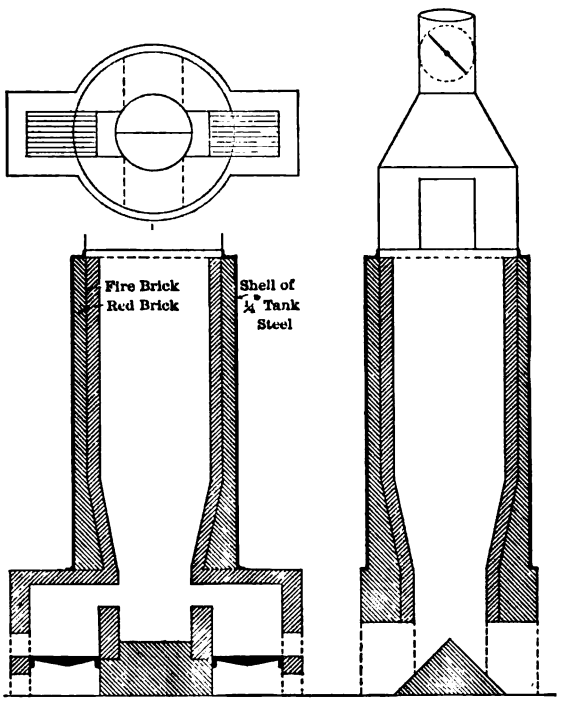
FIG. 5. SHAFT FURNACE WITH ONE FIREPLACE, USED IN SPAIN.
Dimensions in meters.

the circumference, are advisable. It is also desirable to design this type of kiln with closed ash pits, arranging for introduction of the air under the grates through a canal in the masonry of the furnace, by the hot walls of which it will be preheated. A chimney at the top of the shaft, provided with a damper, gives better control of the draught.

Petroleum and Gas Fired Kilns.—I am unaware that calamine is burned anywhere in petroleum or gas fired kilns, although their applicability to that purpose is obvious, inasmuch as they are employed for lime-burning.

In shaft furnaces for burning lime, which are heated by the combustion of petroleum, the fireplaces are replaced by four injectors 90° apart.

The advantage of gas firing for lime-burning kilns, as in other furnaces, lies chiefly in the greater economy that may be realized in fuel because of its more complete combustion to carbon dioxide. It is sometimes claimed that gas fired kilns are less advantageous than petroleum fired, since if the gas is introduced horizontally the flame does not penetrate



FIGS. 6 TO 8. SKETCH FOR SHAFT FURNACE WITH TWO FIREPLACES.

far enough toward the center of the shaft, which consequently has to be made of smaller diameter and correspondingly less capacity, but the same objection would apply to a direct fired kiln with external fireplaces and obviously could be met by introducing the gas at intervals around the shaft.

The Hoffmann ring kiln, which is employed extensively in burning clay products and lime, might undoubtedly be used advantageously in burning calamine if the quantity to be calcined warranted such an expensive installation. This kiln can be fired with slack coal and of all kilns is the most

economical of fuel, only about one third the percentage needed by an ordinary lime kiln being required. This saving is partly offset, however, by the more labor involved in the management of the ring kiln.

Cost of Calcination in Shaft Furnaces.—In burning lime in a shaft furnace about 12% of its weight of carbon is required, equivalent approximately to 15% coal if the latter contains 80% combustible. In burning calamine in a simple furnace, where the coal is charged alternately with the ore from 3 to 10% of coal is required, varying according to the nature of the ore and its admixtures; with externally fired kilns from 6 to 10% is required. The consumption of coal depends largely of course upon its character and calorific power.

In Spain, an externally fired kiln with a single fireplace, having a shaft 13·5 ft. high from the charge opening to the roof of the fireplace, 6 ft. in diameter at the top and 20 in. at the bottom, delivers five to eight metric tons of burned ore per 24 hours, with a coal consumption of 8 to 9% of the weight of the burned ore, the latter losing 20% in calcination. A furnace of the same dimensions, but with two fireplaces puts through 12·5 tons of crude ore, yielding 10 tons of product with a consumption of 0·875 ton of coal and the labor of six men. The cost of calcination per ton of crude ore is consequently that of 0·07 ton of coal and the wages of 0·5 man-shift.

The cost of calcining calamine (smithsonite) at Laurium, Greece, per ton of product is as follows:[1] Fuel (10%, costing 18 fr. per metric ton), 1·80; labor (rate of wages not stated, but probably about 3 fr. per day), 1·10; interest and amortization of cost of furnace, 0·20; superintendence, 0·70; total, 3·80 fr. The ores calcined lose from 25 to 40% in weight; assuming 25%, 1·33 ton of crude ore yields one ton of calcined, or one ton of crude ore yields 0·75 ton of calcined.

Data as to the cost of calcining calamine in shaft furnaces in Sardinia are given in a subsequent section of this chapter.

Reverberatory Furnaces, Hand Raked.—In the more modern plants, shaft furnaces for the calcination of calamine have now been largely displaced by reverberatory furnaces, either hand raked or mechanically raked, which differ in size and some few details, but not in principle, from the reverberatory furnaces for roasting blende. The reverberatory furnaces consume more fuel per ton of ore than the kilns do, and if hand raked more labor is also required; but they are especially adapted to the calcination of fine ore, and even lump ore has been burned in them at some places.

Types and Dimensions of Furnaces.—The calcination of calamine differs from the roasting of blende among other things by the fact that no heat-

[1] Fuchs et de Launay, Traité des Gites Minéraux, II, 386.

evolving reaction takes place, but on the contrary, there is a simple decomposition to effect, for which heat must be supplied. Experience has shown that the hearth of a calamine calcination furnace may be profitably extended to 42 ft. with coal firing, and to 48 ft. with gas firing. The arch should not be more than 24 in. above the hearth at the fire bridge and that distance should be reduced toward the flue, either by elevation of the hearth in steps or on an incline, the floor of the furnace building being made to rise by a corresponding slope so as to maintain a constant elevation of hearth above floor of 3 ft. The width of the hearth is limited, as in blende roasting furnaces, only by the ability of the operatives to work the charge, i.e., to 16 ft. for a furnace with doors on both sides and to half that width for a furnace with doors on only one side. In Europe the calamine burning furnaces are usually of the latter type and the width of hearth is often only 6 ft., which, of course, materially reduces the capacity. The conditions as to working doors and other details of construction are the same in reverberatories for burning calamine as in those for roasting blende, and with respect to these reference should be made to the chapter descriptive of roasting furnaces.

Calamine reverberatories are built generally with one or two horizontal hearths, and sometimes with a single inclined hearth. Both direct and gas firing are used.

Calcination Kilns Connected with Distillation Furnaces.—It was formerly a common practice to connect a small calcining hearth with a distillation furnace, utilizing the waste heat of the latter for burning the calamine, but this practice has been generally abandoned as interfering with the draught of the distillation furnace and otherwise disturbing its more delicate operation, although furnaces thus arranged are still to be seen at Lipine in Upper Silesia, where there is one of the largest smelteries in that Province. Between each double distillation furnace at those works there is placed a calcining hearth 2·2 m. long, 0·8 m. wide, with 0·7 m. height from hearth to roof. In 24 hours from 2·25 to 3 metric tons of ore are burned, the operation being conducted by the crews of the distillation furnaces; since the heat is also furnished by the distillation furnaces the cost of calcination in this case is insignificant.

Examples from Practice.—At the Paulshütte near Rosdzin, in Upper Silesia, a hearth 5·45×2·4 m. burns 15 tons per 24 hours with a consumption of 10% of coal, loss in weight not stated.[1]

A reverberatory furnace employed long ago at Altenberg (Moresnet) had two hearths, of which the lower was 5 m. long and the upper 7·2 m., a total

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 30.

length of 12·2 m. (approximately 40 ft.). The lower hearth was 1·5 m. (5 ft.) wide at the bridge, 2·92 m. (9 ft. 9 in.) in the middle and 1·46 m. (4 ft. 10 in.) at the flue communicating with the upper hearth. The grate was 0·6×1·5 m.; it was 0·5 m. below the top of the fire bridge, which was 0·35 m. thick and 0·25 m. above the hearth. The arch over the lower hearth was 0·4 m. above the latter. The flue leading from the lower hearth room to the upper was 0·38×1·46 m., and that leading from the upper hearth to the chimney was 0·5 m. wide. This furnace calcined 8000 kg. of calamine per 24 hours, which quantity was fed in four charges, each charge remaining six hours on the upper hearth and six hours on the lower. The loss of weight in calcination was 27 to 30%.

MECHANICAL REVERBERATORY FURNACES.—Any of the mechanically raked reverberatory furnaces which are employed for blende roasting are undoubtedly applicable to the calcination of calamine, but since this work is done usually at the mines there are few which make individually so large an output as to warrant so expensive an installation. There are, however, several works, especially in Sardinia, where revolving cylinders are in use, the Oxland & Hocking, which is shown in Figs. 19 and 20, being the common type. Any of the other standard cylinder furnaces, e.g., the Brückner, would doubtless be applicable to this work. The Argall four-cylinder furnace would appear to be especially well designed for it.

CALCINATION OF CALAMINE IN SARDINIA.

The calcination of calamine is practised in Europe to much greater extent than in America. The ore of this class which is mined in Algeria, Greece, Italy and Spain, is generally calcined at the mines before shipment. The most important producers of calamine are situated on the Italian island of Sardinia, and the methods employed by them for the calcination of calamine are typical of the best practice in Europe. Signor Erminio Ferraris, director of the mines and works of the Societa di Monteponi at Monteponi, Sardinia, has kindly aided me in the preparation of this treatise by furnishing a description of the methods employed in Sardinia, of which the following is a translation:

CALCINATION IN SMALL SHAFT FURNACES (KILNS).—At the small mines and in provisional installations the old type of shaft furnace, from 4 to 5 m. in height and 2 m. in diameter inside, is still employed. The shaft of these furnaces is now made cylindrical in form, but some very old furnaces are still to be found, which are greater in diameter in the middle than at the top and bottom, having somewhat the shape of a barrel, as shown by the dotted lines in Fig. 9.

These furnaces are generally constructed of sandstone and Silurian schist, which withstand sufficiently well the heat required for the calcination of the ore; sometimes, however, the shaft is lined with brick. At the bottom the furnace has generally four openings of conical shape for discharging the burned ore and a brick pyramid in the center of the hearth to direct the calcined ore toward the openings. The latter are closed by movable doors of wood on the leeward side and also on the windward side if the wind be too strong.

The calamine is charged in layers of 0·15 to 0·20 m. in thickness, care being taken to cover each layer with fine ore. Upon the layer of ore is thrown a thin layer of fuel. The furnace is charged thus with successive

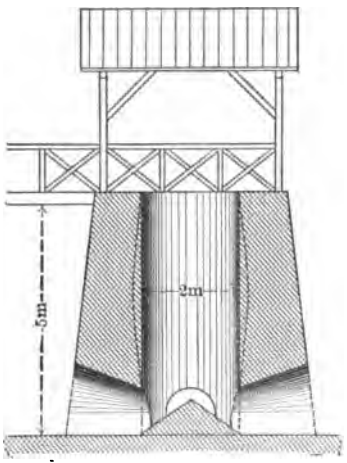


FIG. 9. OLD FORM OF SHAFT FURNACE, USED AT MONTEPONI, SARDINIA.

layers up to the top. The fuel generally employed in these low shaft furnaces is small charcoal. For the carbonates of zinc the quantity of charcoal required is from 7 to 10% of the weight of the ore, varying according to the quality of the charcoal and the moisture in the ore; for silicate ore from 4 to 6% of charcoal is used.

The calcined ore is drawn once or twice in 24 hours, the proper time for drawing being indicated when it is perceived that the fire has risen nearly to the top of the charge in the furnace. If much fines are used with the lump calamine, the calcination proceeds more slowly than when only large pieces are charged. The rate of calcination is consequently governed to a great extent by the proportion of fines in the charge, and may range from two tons to eight tons of calcined product per 24 hours.

The calcination is generally carried to the point where an assay of the product by heating a sample in a clay crucible shows a further loss of weight of not to exceed 3%. The large lumps of ore are broken to the size of the fist, not more. At the small mines where there are not special furnaces for the calcination of fines and slimes below 15 mm. size, the calcination of from 10 to 20% of fines with the lump ore is accomplished, but much ability is required in managing the furnace to prevent the fire from being extinguished. It is always necessary, however, to spread a little fine ore on each layer of lump ore in order to support the charcoal and prevent it from working its way down through the underlying layers.

LARGE SHAFT FURNACES.—At the Monteponi, Malfidano and some other mines, the shaft furnaces are installed in blocks, which are so arranged that the ore can be drawn from two sides. The arrangement is shown diagrammatically in Fig. 10. The furnaces at Monteponi have shafts of 6 m. in height, slightly conical in shape, tapering toward the top (the walls having an inclination of 3% from the vertical). This interior contour has been found to be the one which permits the charge to descend most uniformly,

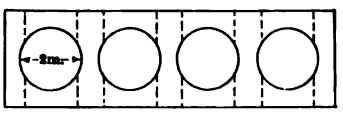
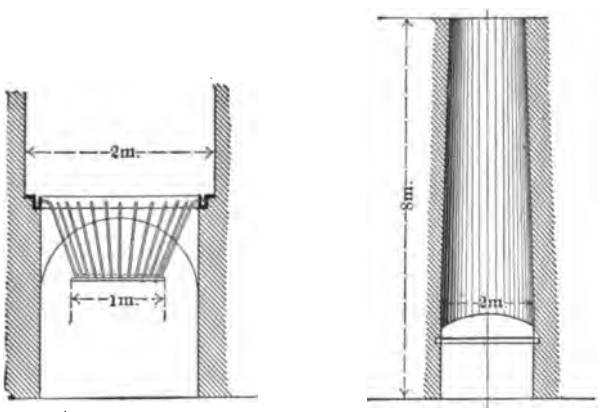


FIG. 10. ARRANGEMENT OF SHAFT FURNACES IN A BLOCK.

preserving always a horizontal surface, and the furnaces so designed have taken the place of the old type with the belly in the middle. The furnaces at Monteponi have at the bottom of the shaft a conical, basket-like grate of round iron bars supported at the top and bottom by rings bent from flat iron bars, as shown in Fig. 11. The horizontal portion of the grate at the bottom consists of short bars inserted between those of the basket. The side bars of the basket are from 16 to 20 mm. in diameter; the loose, horizontal bars are from 30 to 40 mm. In order to draw the charge the horizontal bars are turned, causing the ore to drop between them into a car standing beneath them, unless it be that the bars are placed very close together, when the ore is dropped by drawing out some of the bars. At Malfidano the furnaces have at the bottom of the shaft a horizontal grate made of loose round iron bars which rest upon cast iron bearing bars, as shown in Fig. 12.

The system of charging these large shaft furnaces is the same as with the small ones, except that a different fuel is employed, a mineral fuel, such as lignite slack, lean coal, coke fines and anthracite, being used preferably in

the large furnaces. Anthracite gives the best results with respect to consumption and with it the proportion of fuel may be as little as 3% for silicates which are free from hygroscopic moisture, but very delicate management of the furnace is required to accomplish that result. The fuel which is the most easily employed is lignite slack, smaller than 10 mm. and greater than 5 mm. in size. To start a furnace on a new campaign a good wood fire is made on the grate, or on the hearth if there be no grate, and the combustion being well under way, the charging is begun as described under the caption of "Calcination in Small Shaft Furnaces." If the fire be permitted to go out in a furnace, it is not necessary to empty it in order to start anew, but a fresh wood fire is made on top of the ore



FIGS. 11 AND 12. SHAFT FURNACES WITH GRATES.

The shaft furnaces of this type used in Sardinia are 8 m. in height from the ground to the top; 1·7 m. in diameter at the top and 2 m. at the bottom.

remaining in the furnace, and when it has been well kindled a fresh charge is thrown on and the regular procedure is then followed.

With large, modern shaft furnaces running normally, well attended to and not charged with too great a proportion of fine ore, calamine can be calcined in Sardinia at a cost of 3 fr. (60c.) per metric ton of raw ore charged, the cost of the fuel being 20@25 fr. ($4@$5) per metric ton for small charcoal, 15 fr. ($3) for lignite and 30@40 fr. ($6@$8) for anthracite. The men employed in the calcination process earn an average wage of 3 fr. (60c.) per day, working 12 hours.

REVERBERATORY FURNACES.—The old reverberatory furnaces had two superimposed horizontal hearths, each 7 m. in length. Some of these still exist in Sardinia, but in general they have been abandoned, being too

costly and with the ordinary direct firing unable to calcine the limestone which accompanies the calamine. This type of furnace has been displaced therefore by a special form of gas fired reverberatory furnace and by revolving cylinders.

At Monteponi a gas fired reverberatory furnace of peculiar design has been in use since 1885.[1] At the time it was built, three Oxland furnaces were being erected at the Bugerru mine in Sardinia and there was considerable discussion among Italian metallurgists as to the relative economy of the two types. The experience has proved, however, that the Oxland

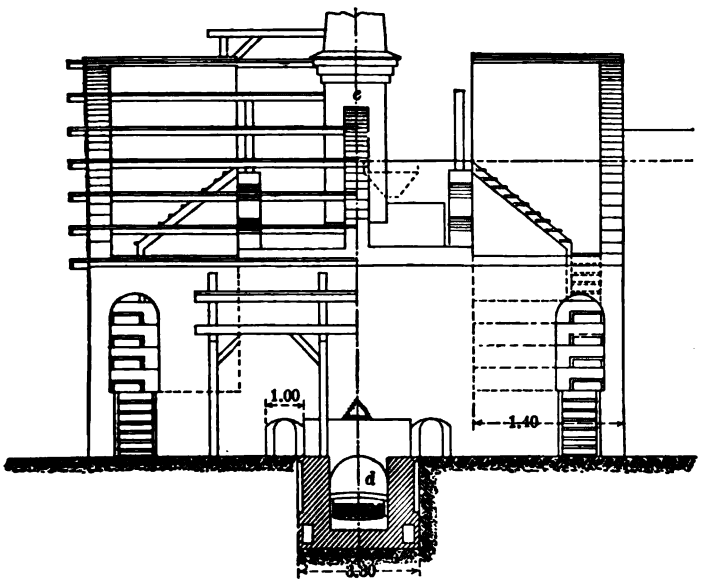


FIG. 13. FERRARIS REVERBERATORY FURNACE.
End elevation. Dimensions in meters.

furnaces have no advantage of economy in operation, while the repairs required on them are more costly and inconvenient than on the Ferraris furnace.

The Ferraris furnace has a steeply inclined hearth *a*, divided into two sections by a central longitudinal wall. At the upper end of each section there is a cast iron charging hopper *b* and at the lower end a discharge opening *c*. The two sections of the hearth have a common fireplace *d* and a common chimney *e*. On each side of the furnace there are five working doors *f*.

The fire box, or gas producer, is arranged with step and plane grates

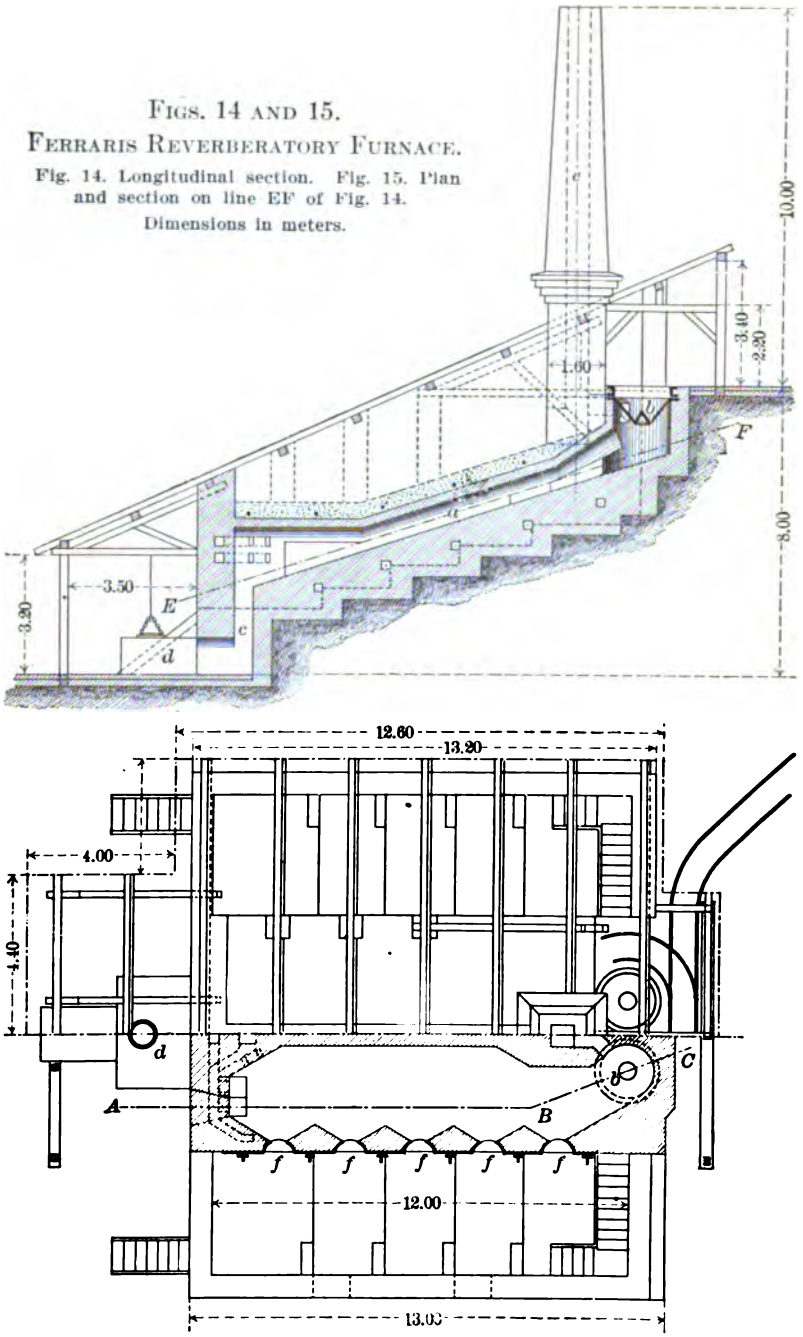
[1] This furnace was designed by Signor Ferraris, by whose name it is commonly known.

FIGS. 14 AND 15.

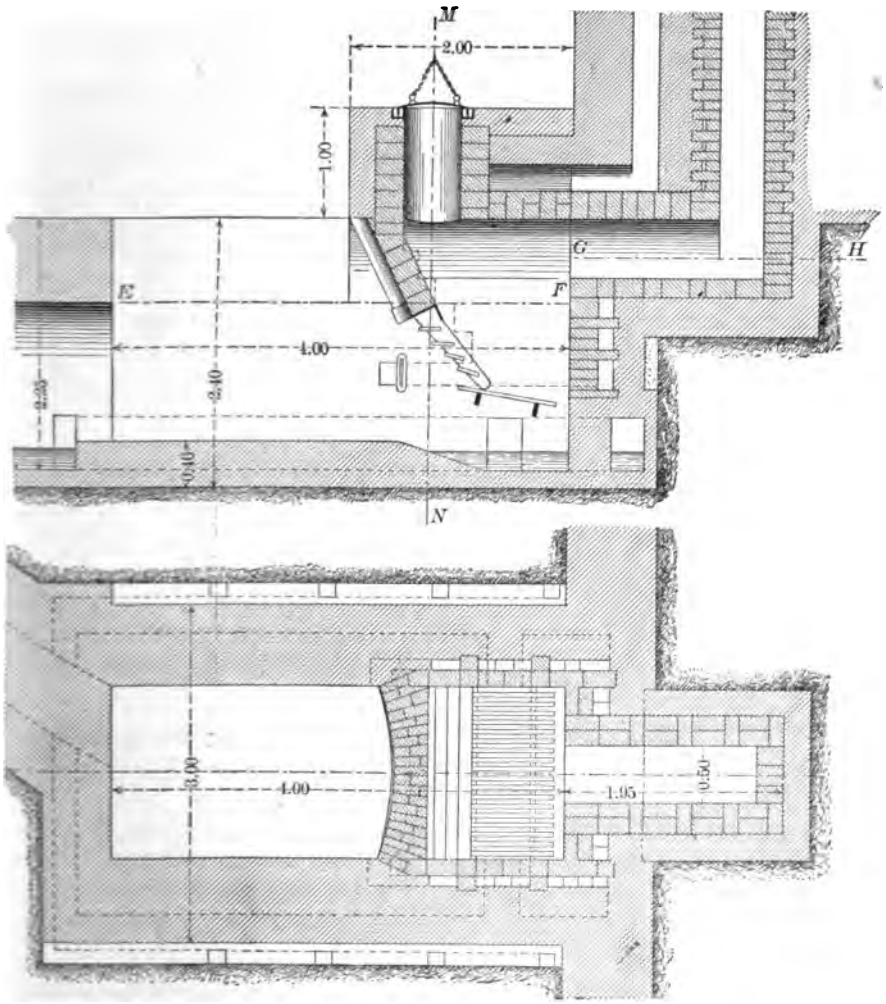
FERRARIS REVERBERATORY FURNACE.

Fig. 14. Longitudinal section. Fig. 15. Plan and section on line EF of Fig. 14.

Dimensions in meters.



an air space being left between the two parts, which are bonded by projecting bricks here and there. Air is admitted into the space between the two walls, the supply being controlled by a suitable damper, and having cooled



FIGS. 16 AND 17. GAS PRODUCER OF FERRARIS REVERBERATORY FURNACE.
Fig. 16. Longitudinal section. Fig. 17. Horizontal section on EFGH of Fig. 16. Dimensions are in meters.

after the type of the well known Boëtius generator. The interior chamber is built of fire brick and is enclosed within a larger box of common brick,

the inner wall and thereby itself having been heated, it passes into the lower part of the calcination chambers, where it burns the gas streaming in from the generator.

The lower part of the calcination chamber *a* is made large, being covered by a horizontal arch, in order that the ore to be calcined may be heated by radiation rather than by contact. The chamber is then contracted, the roof arch being inclined parallel with the hearth, in order that the products of combustion may give up more of their heat by contact with the calamine. At the upper end the chamber is again enlarged around the charge hopper *b*, in order that the gases before reaching the chimney may heat the fresh

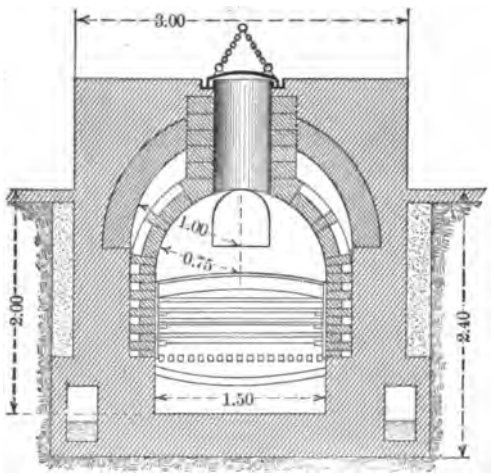


FIG. 18. GAS PRODUCER OF FERRARIS REVERBERATORY FURNACE.

Transverse section on line MN of Fig. 16. Dimensions are in meters.

ore contained in the hopper. The charge hoppers may be provided with a cast iron cone to regulate the admission of the ore, or the latter may be allowed to fill the upper end of the calcination chamber up to the hopper, the cone being omitted, whereby the feed is regulated according to the manner in which the ore is manually moved forward on the hearth.

The calcination chamber is divided into two sections in order that the gas production may not be interfered with when the working doors on one side are opened. The effect of the division into two sections is that all the doors on one side may be open without disturbing the action of the generator, since when that side is cooled off thereby, the other side, heated more strongly, produces a stronger draft.

The cost of installing such a double furnace is about 20,000 fr. ($4000), of which about 5000 fr. ($1000) is for the excavation and shed and 15,000 fr. ($3000) for the furnace itself.

A double furnace of the dimensions shown in the accompanying engravings will produce monthly from 580 to 650 metric tons of calcined zinc silicate ore with a coal consumption of 9·8 to 10% of the weight of the calcined product. This coal consumption is attained with the use of lignite having a heat value of 4800 calories per kilogram of combustible (hygroscopic water and ash being deducted).[1] The figures of coal consumption above given represent the difference between the weight of the fresh lignite used and the cinder or coke which is drawn from the generator and is employed subsequently in the calcination of calamine in shaft furnaces. The quantity of fresh lignite charged into the generator is about double the percentage stated above.

The calamine which is calcined contains a certain proportion of blende, which may rise as high as 10%, together with a small tenor of galena. The sulphur of those minerals is almost entirely removed during the calcination. The size of the ore has practically no influence upon the result, since pieces of calamine free from sulphur can be calcined as completely when of the size of the fist as can the fine slimes from the dressing works.

The cost of calcination per 1000 kg. of product amounts to 2·20 fr. for labor and 1·20 fr. for fuel, the latter costing 12 fr. per metric ton, making the total 3·40 fr. per metric ton of product. If, however, the consumption of fuel is reckoned on the basis of the quantity which is introduced into the producer, without any deduction for the coke, the expense for fuel is 2·40 fr. and the total 4·60 fr. per metric ton of calcined product. If the calamine is compact zinc carbonate the cost of calcination is greater, since it is impossible to put through so large a quantity per month as reported above, those figures representing the results with calamine in which zinc silicate predominates, without admixture of limestone or dolomite. Practically with hard zinc carbonate ore it is necessary to reckon on a consumption of 20% of fuel and an expense of 5@6 fr. per metric ton of calcined product.

Revolving Cylinder Furnaces.—Revolving cylinder furnaces of the Oxland type are employed to a considerable extent in Sardinia for the calcination of ore fines up to 30 mm. in size. They are not adapted to the calcination of calamine between 15 and 30 mm., since they consume more fuel than the shaft furnaces, but they are able to calcine unsized fines up to a maximum of 30 mm. pieces, which can be handled only with difficulty in the shaft furnaces, as has been stated previously.

[1] The lignite employed in Sardinia contains 6% sulphur, 10% ash and 6% hygroscopic water.

The excellent furnaces which are installed in the magnetic ore separation plant at Monteponi and are illustrative of the type are shown in Figs. 19 and 20. The ore is received in the charging hoppers, *a*, from which it is conducted into the cylinder through a tube 200 mm. in diameter. Each furnace consists of a cylindrical wrought iron shell, 10 m. in length, lined with wedge-shape fire brick, forming a cylindrical calcination chamber 1 m. in diameter. The shell is girdled by four steel bands which rest upon eight wheels. It is turned by means of a worm gear at the rate of 16 revolutions per hour. The feed of the ore is regulated by adjustment of the tube leading from the charge hopper into the cylinder, by varying the distance between its end and the lining of the cylinder. In general the ore is six hours in traveling the length of the furnace. The calcined ore falls between the

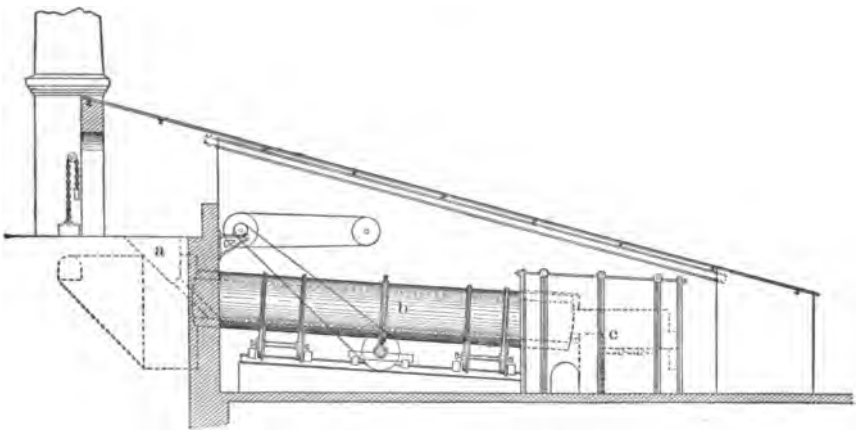


FIG. 19. ELEVATION OF OXLAND-HOCKING FURNACE.

lower edge of the cylinder and the fire box into a closed chamber, where it is permitted to cool somewhat before being removed.

The fire box is of the ordinary construction, being a fire brick chamber covered with an arched roof and provided with a flat grate, $1{\cdot}5 \times 0{\cdot}5$m.$=0{\cdot}75$ sq. m. in area, and an undergrate blast. A rather thick bed of fuel is carried on the grate, so that the combustion is incomplete and only partially burned gas is developed, the combustion of the latter being completed in the calcination chamber (cylinder). The air necessary for the combustion enters through two openings to the right and left of the fire door into a chamber which surrounds the arch of the fire box, wherein it is heated by contact with the arch and thence passes through two slits in the fire bridge into the interior of the cylinder, producing a sharp flame with the

unburned gases of combustion coming over from the fire box. The fire bridge is built in tubular form of wedge-shape brick and projects into the cylinder, whereby the edges of the latter are protected from the flame. The air which is drawn in through the crevice between the fire box and the cylinder is sufficient to cool the masonry near the fire bridge.

Furnaces of the above type are employed at all the zinc mines of Sardinia, with slight differences in dimensions. In general the length varies from 8 to 12 m. and the exterior diameter from 1 to 1·3 m. The fire is

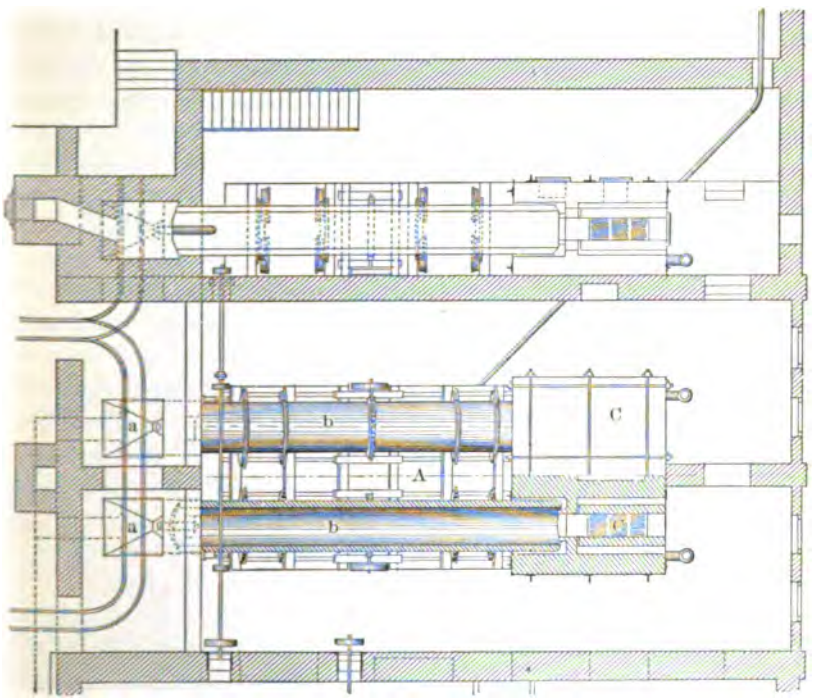


FIG. 20. PLAN OF OXLAND-HOCKING FURNACE.

everywhere blown by means of a Koerting air injector, and the air for secondary combustion is preheated around the walls of the fire box, thus constituting a species of semi-gas-firing. Lignite is employed as fuel, which gives 30% of gas, and on the grates, blown with steam, burns almost completely, a result which is not attained in the gas producers which work by natural draft. It is necessary to reckon a consumption of 20% of lignite to obtain a good calcination with hard ores. There are two Oxland furnaces in Sardinia which are heated with wood and give good results.

III.

BLENDE ROASTING.

In roasting zinc blende, preliminary to distillation of the zinc, the aim is to convert the sulphide as completely as possible into oxide, since every part of sulphur remaining in the roasted ore means approximately two parts of zinc held back in the retort residues after distillation. The accomplishment of this object requires fine comminution of the blende, generally not to exceed 2 mm. size (i.e., about 0·08 in., or such as will pass a standard six-mesh wire cloth), high temperature, and slow and careful roasting, with frequent stirring. It is never economical to effect a complete elimination of the sulphur, which is anyway impossible with some ores, but in good practice the amount in combination with zinc, iron and lead is reduced to 1% and frequently lower; ores containing lime and magnesia may retain much more, owing to the formation of calcium and magnesium sulphates. Calcium sulphate is formed by the action of sulphurous anhydride on calcium oxide; it is not decomposable at the temperature attained in the roasting furnace. Magnesium sulphate is formed in an analogous manner, but it is a less stable compound than calcium sulphate and is partly decomposed in the last stage of the roasting process.

Chemical Reactions in Blende Roasting.—The reactions which take place in roasting blende have been described in detail in numerous metallurgical treatises, especially by Plattner in *Die Metallurgischen Röstprozesse* (1856), who first laid down the principles, and for a full discussion of the chemical changes which are involved reference should be made to that treatise and to Balling's *Metallurgischen Chemie* (1888). Expressed briefly, the zinc sulphide begins to be oxidized, or begins to burn, at dull red heat; as the temperature increases to bright red the oxidation proceeds more energetically, with the formation of zinc oxide and sulphurous anhydride, the ultimate result being represented by the equation:

$$ZnS+3O=ZnO+SO_2.$$

The sulphurous anhydride escapes partly as such, but through contact with the glowing particles of ore and the hot furnace walls it partly takes up another atom of oxygen (by contact action or catalysis) and the sulphuric anhydride thus formed combines with zinc oxide as neutral zinc sulphate ($ZnSO_4$). At cherry red heat the neutral zinc sulphate is decomposed into basic sulphates ($3ZnO,ZnSO_4$, or analogous compounds) and sulphuric anhydride, the latter being in turn more or less completely broken up into sulphurous anhydride and oxygen. The basic zinc sulphate is not decomposed entirely until it has been exposed to bright red heat for a considerable time, wherein there is, when reverberatory furnaces are employed, the concomitant danger of loss of zinc by volatilization, probably for the most part through reduction of zinc oxide by carbon monoxide in the fire gases, since zinc oxide itself is not dangerously volatile until white heat is reached.[1] The high temperature required for the decomposition of basic zinc sulphate involves the further hazard of a partial sintering of the charge when lead, iron and manganese are present, and a loss of zinc through particles still undesulphurized becoming enveloped in the fusible slag.

In roasting as a preliminary to the treatment of mixed sulphide ores, it is sometimes considered desirable to produce the maximum possible amount of neutral zinc sulphate. This is accomplished by roasting, necessarily slowly, at a very low temperature (never higher than dull red). The formation of the sulphate is favored by the presence of the sulphides of other metals than zinc, especially by the sulphides of iron.[2] This results from the fact that the monosulphide of iron (whether occurring as such isomorphous with the blende, which is frequently the case, or as pyrite or marcasite, which lose the extra atom of sulphur at a comparatively low temperature and become monosulphide) is oxidized more quickly than the zinc sulphide. The FeS becomes changed to $FeSO_4$ or Fe_2SO_6; these are decomposed, liberating SO_3, which attacks ZnO and forms $ZnSO_4$, which can exist at the temperature whereat the sulphates of iron are broken up. Hence the presence of iron in the ore favors the formation of zinc sulphate.

There is ground for the belief that the formation of sulphuric anhydride in the roasting furnace is not due merely to the oxidation of sulphurous anhydride, but on the contrary that it, and not sulphurous anhydride, may

[1] According to Stahlschmidt (Berg- u. Hüttenm. Ztg., 1875, p. 69) zinc oxide is appreciably volatile at the melting point of silver (970° C.), about 15% more so at the melting point of copper (1054° C.), and rapidly at white heat (1150° to 1200° C.). Roasted blende is not volatile at 970° C., but is considerably at 1054°.

[2] In roasting mixed blende and galena, from Broken Hill, N. S. W., Schnabel succeeded in converting 40% of the zinc into neutral sulphate; in the treatment of Algerian ore by the Parnell process at Swansea, in 1879 to 1883, about 33% of the zinc was so converted regularly.

be the primary product of the oxidation of the sulphur. It is a general rule of thermo-chemistry that when two or more compounds are possible, that one will be formed which produces the greatest heat in the reaction; for example, carbon and oxygen can form carbon monoxide or carbon dioxide, of which the latter is the usual product of complete combustion. Sulphur and oxygen can form sulphurous anhydride and sulphuric, but the former is the usual product, or rather the usual ultimate product. The reason for this is perhaps that the heat of combination of sulphuric anhydride is so great as to raise the temperature beyond the point where that compound is stable, wherefore it is dissociated into sulphurous anhydride and oxygen. Similarly in the combustion of carbon, when the temperature exceeds a certain limit carbon monoxide is produced instead of the dioxide.

Many facts point to the probability that the first product of combustion of sulphur is sulphuric anhydride, which is subsequently broken up. It is well known that in burning pyrites the gas contains proportions of sulphuric anhydride varying from 2 to 20% of the total sulphur oxides, which fact has been generally explained as being due to the presence in the furnace of ferric oxide and its catalytic action on the gas, but the undoubted presence of sulphuric anhydride in the gas obtained from the combustion of brimstone cannot be accounted for in that way, and there seems to be a probability that if means could be found to get rid of the excess of heat of combustion in the furnace itself, the direct product of such combustion would be SO_3.[1] That idea is supported by the fact that in burning pyrites the percentage of sulphuric anhydride in the gas is highest when the burners are in bad condition and cool; also by the fact that in roasting blende the percentage of zinc sulphate that is formed is greatest when the temperature is lowest.

Practically the formation of zinc sulphate appears to be increased also by roasting in a mixed atmosphere of air and steam. The proposal to desulphurize blende by roasting in an atmosphere of steam alone, according to the reaction

$$ZnS+H_2O=ZnO+H_2S$$

is without practical value, because of its incompleteness even at a high temperature.

Decomposition of Zinc Sulphate.—Zinc sulphate is dissociated by heat into zinc oxide, sulphuric anhydride, sulphurous anhydride and oxygen, but complete decomposition is effected only at a high temperature, more specifically at a bright red heat. When the temperature is raised only to cherry red, neutral zinc sulphate, $ZnSO_4$, is split up into basic zinc sulphates and

[1] F. J. Falding on The Manufacture of Sulphuric Acid, in The Mineral Industry, VIII, 589.

sulphuric and sulphurous anhydrides and oxygen. There is no record in chemical or metallurgical literature, so far as I have been able to discover, as to which of the several basic sulphates of zinc are produced during that process, except a remark in Plattner's *Röstprozesse* that it is the tetrabasic sulphate ($ZnSO_4,3ZnO$) which is formed.[1] It is not unlikely, however, that other basic sulphates are produced in the course of the process. The basic sulphates of zinc are not in any case completely decomposed until the high temperature mentioned above is attained.

The formation of oxygen and sulphurous anhydride in decomposing zinc sulphate is due to the temperature at which the process is performed. If the decomposition could be effected at a low temperature, the gas evolved would be pure sulphuric anhydride, but that gas being unstable at high temperatures is split up into oxygen and sulphurous anhydride.[2] There is nevertheless a considerable proportion of the sulphuric anhydride, probably to the most extent that which is first driven off, which escapes decomposition. Schlapp found, in desulphurizing zinc vitriol (dehydrated as completely as possible) on a large scale at Broken Hill, that about 30% of the sulphuric anhydride escaped as such.

The proposal to facilitate the desulphurization of zinc sulphate by roasting with an admixture of carbon, taking advantage of the reaction

$$ZnSO_4+C=ZnO+SO_2+CO,$$

has not yet proved to be of any practical importance, although the decomposition in that manner takes place at a lower temperature than the simple dissociation of zinc sulphate. The reduction of zinc sulphate to oxide by means of carbon takes place, indeed, only at a comparatively low temperature. At high temperatures zinc sulphate is reduced by carbon to sulphide, which is what happens when zinc sulphate is present in the charge undergoing distillation in the retort.

Effect of Impurities in Blende Roasting.—It will be observed from the foregoing that although the chemical behavior of zinc sulphide when heated in the presence of oxygen (air) is understood in so far as the practical results are concerned, there is not even yet a thorough knowledge of the purely chemical reactions which take place in an apparently so simple process; on the contrary, many deductions have been drawn from incom-

[1] Die Metallurgischen Röstprozesse, p. 142.

[2] The matter of temperature has been found to play a very important part in the contact process of sulphuric acid manufacture. From 380° to 430° C. nearly all of the sulphur dioxide is converted to trioxide; above 430° the action diminishes, and at 700° to 750° the transformation is only about 60%. At 900° to 1000° C. action between sulphur dioxide and oxygen can no longer be detected. (R. Knietsch, Berichte der deutschen Chem. Gesell., 1901, XXXIV, xvii, 4069 to 4115).

plete experimental data, in this as in other pyrometallurgical processes, and further experiments are needed with the employment of modern pyrometrical facilities and methods.

Thus it is to be pointed out that all blendes do not behave the same in roasting, some varieties being desulphurized more easily than others. Jensch, an experienced German metallurgist (lately deceased), remarked that blendes from the older geological formations of Scandinavia gave up their sulphur less readily than those from the later formations of Styria (Austria).[1] No scientific reason can be assigned why that should be so. Dark colored blende, high in iron, is more difficult to roast than are the light colored varieties.[2] Jensch found that the sulphur remaining in a ferruginous blende after roasting was present as iron sulphide, not zinc sulphide, which is surprising, since iron sulphide is more easily oxidized than zinc sulphide, while ferrous sulphate is dissociated at about 590° C.

Prost, on the other hand, has shown[3] that both iron and zinc sulphides are almost completely converted to oxides, while lead sulphide becomes lead sulphate and silicate, and calcium and magnesium form sulphates. This is the result one would expect in the light of our present knowledge, since the decomposition of lead sulphate, it is agreed, requires a higher temperature than zinc sulphate, a white heat liberating sulphuric anhydride only to a small extent and a stronger acid, such as silica, being needed to expel it completely from the combination.[4] In a sample of roasted ore containing 57·80 units of zinc, only 0·20 unit was present as sulphate, while out of a total of 5·60 units of lead 5·02 were in that form.

Prost also considers that there is evidence that ZnO enters into combination with Fe_2O_3, forming zinc ferrate, $ZnFe_2O_4$. The formation of zinc ferrate might explain why in leaching a thoroughly roasted ore with sulphuric acid the extraction of zinc is so incomplete, as has been found to be the case in numerous processes proposed for the treatment of mixed sulphide ore in that manner. Three samples of ore digested by Prost with ammonium tartrate gave the following results:

Number.	Iron.	Zinc.	Zinc Soluble in Amm. Tart.	Proportion Sol. in Amm. Tart.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I.....	15·20	45·29	28·82	63·64
II.....	1·50	62·86	59·18	95·42
III.....	5·28	57·80	43·16	74·67

[1] Berg- u. Hüttenm. Ztg., 1894, 299; Zts. f. angew. Chem., 1894, p. 50.

[2] Minor, Berg-u. Hüttenm. Ztg., 1889, p. 466.

[3] Bull. de l'Assoc. Belge des Chimistes, X., vi, 246 to 263.

[4] Hofman, Met. of Lead, 5th Ed., p. 150.

Calcite and Barite.—In roasting a zinc ore which contains calcite, the latter is converted partly into lime and partly into calcium sulphate, wherefore it is impossible to desulphurize completely a calcareous blende. The presence of calcium sulphate in the roasted ore used to be considered objectionable in the distillation process, inasmuch as it is reduced to sulphide by the carbon in the charge and calcium sulphide was thought to react with zinc, forming zinc sulphide, and thereby increasing the tenor of zinc in the retort residue, but although this opinion is still held by many metallurgists careful experiments (vide Chapter VI) have proved it to be incorrect.[1] Barytes remains unaltered in the roasting furnace, but like calcium sulphate in the distillation furnace it is reduced to barium sulphide, which probably has an equally harmless effect.

Dolomite.—Dolomite behaves in roasting somewhat like calcite, but it is more objectionable in zinc ore, since magnesium sulphate is probably to some extent reduced by carbon in the retort to magnesia and sulphurous anhydride and favors the production of zinc sulphide.[2] Fortunately magnesium sulphate, unlike calcium and barium sulphates, is decomposed to a considerable extent in the roasting furnace.

Galena.—The presence of galena in zinc ore which is low in silica leads to retention of sulphur in roasting because of the formation of lead sulphate, which can be decomposed only by the action of silica. Even if there be sufficient silica present for that purpose the presence of lead is still undesir-

[1] According to Thum (Berg- u. Hüttenm. Ztg., 1876, p. 154) calcium sulphate reacts with zinc in the presence of carbon as follows:

$$2CaSO_4+Zn+4C=Ca_2S+ZnS+4CO_2$$

In Kerl's Grundriss der Metallhüttenkunde, p. 431, the following reaction is expressed to account for the formation of zinc sulphide in the presence of calcium sulphide, which is produced by reduction of calcium sulphate by carbon:

$$CaS+ZnO=ZnS+CaO.$$

According to Berthier, however, zinc sulphide itself is reduced by lime in the presence of carbon, calcium sulphide and zinc being formed. (Tr. des Essais, II, 570). The completeness of the reaction is stated as being dependent upon the temperature; in heating 6·32 g. $CaCO_3$+6·03 g. ZnS to a very high temperature, five sixths of the zinc was volatilized. The tendency of lime to decompose zinc sulphide in the retort has been determined by Professor Eugène Prost in a series of careful experiments, who found moreover that calcium sulphate at 1200° to 1250° C. exerted no retarding influence on the reduction of zinc oxide even in the presence of lead and iron silicates, which might convert reduced calcium sulphide to silicate and thus facilitate the formation of zinc sulphide (Bull. de l'Assoc. Belge des Chimistes, X, vi. 246 to 263.) Doctor Orgler, an experienced Rhenish zinc smelter, was also of the opinion that the reaction

$$ZnS+CaO=ZnO+CaS$$

took place in the retort, during distillation (Wagner's Jahresberichte, 1890, 415). Recently Karl Sander, of Prayon, Belgium, has contributed further experimental data upon this subject (Berg- u. Hüttenm. Ztg., 1902, LXI, 465). He charged 10 retorts with roasted blende from Southern France, which contained lead and 9·2% of barytes. He charged 10 more retorts with a roasted blende which was free from barium. The residua from the former assayed 2·08% Zn; those from the latter 2·92% Zn. A repetition of the experiment gave 2·70% and 3·54% Zn respectively.

[2] Prost, Bull. de l'Assoc. Belge des Chimistes, X, vi, 246 to 268.

able, since the fusible silicate of lead may envelop particles of blende and prevent their desulphurization. In roasting a lead-bearing ore there is a considerable loss of lead by volatilization, which may amount to 20%. An experiment with an ore assaying 26% Pb and 36% Zn showed a loss of 6·5% of the lead. Experiments by K. Sander in roasting ores containing 9·84%, 11·38%, 6·60%, 7·73% and 6·00% Pb showed losses of lead of 9·53%, 7·88%, 16·44%, 19·05% and 21·80% respectively.

Mercury.—This element is not often associated with zinc, but some blende was once treated at Oberhausen, Westphalia, which contained 0·02% Hg. The mercury was of course volatilized in roasting the blende; it was condensed in the dust chambers, the dust collected assaying 7% Hg, and was recovered therefrom.[1] Sludge collected from the sulphuric acid chambers assayed 4% Hg. These results were exceptional. According to Mr. Ross, director of the works at Oberhausen (Société Anonyme de la Vieille Montagne), the flue dust ordinarily contains only 0·01 to 0·02% Hg.[2]

Fluorine.—According to Professor Prost fluorine is of common occurrence in zinc blende ores. Generally it exists only in traces, but in some varieties it amounts to several per cent.[3] One sample showed 7% F. In roasting such ores the fluorine, whatever be its form in the furnace gases, is in all probability converted into fluorhydric acid when it comes in contact with the sulphuric acid in the Glover tower and is very injurious to the lead chambers. The harmful effect is entirely out of proportion to the percentage of fluorine, since even small traces, besides being injurious in themselves, accelerate the corrosion of the lead by sulphuric acid and nitrogen compounds. The corrosive effect of gaseous fluorhydric acid was tested in a small apparatus in which the reactions of the lead chambers were duplicated. Before admission of the fluorhydric acid gas, the loss of lead was 7·825 g. per sq. meter; after the introduction of 2·05 g. of HF (a very small quantity in proportion to the total gases) the loss increased to 141·58 g.[4]

In a subsequent series of experiments, Professor Prost found that in roasting a blende containing 7% F, 8·37% SiO_2 and 0·62% Fe, ground very fine, 93% of the fluorine was driven off at red heat. A sample containing 2·02% F, 4·66% SiO_2, and 0·44% Fe showed an elimination of 31% of the total fluorine, whether ground coarse or fine. A blende containing 2·30% SiO_2 and 10·30% Fe, to which 10% of fluorine was added, ground to 2 mm. size, lost 78% of the fluorine, which was increased to 90% by fine grinding. Finally a blende containing 14·75% SiO_2 and 5·48% Fe, to

[1] Chem. Ztg., X, 1039.

[2] Rev. Universelle des Mines, 1894, XXV, 47 et seq.

[3] Near Marion, Ky., there are veins of zinc ore which is an intimate mixture of blende and fluorspar.

[4] Bulletin de l'Assoc. Belge des Chim., Dec. 12, 1901; Chem. Ztg., 1902, XXVI, 1, 12.

which 10% of fluorine was added, lost the whole of the fluorine when finely ground and roasted, but only 89% when coarsely crushed. There appears to be some relation between the percentage of silica, fineness of subdivision and proportion of fluorine driven off. Professor Prost is of the opinion that at least a portion of the fluorine is liberated as free fluorhydric acid.[1]

ARGENTIFEROUS BLENDE.—Many varieties of blende received by the zinc smelter are argentiferous, and sometimes the silver content of the charge distilled is sufficient to make it worth while to smelt the residuum, either directly or after a preliminary mechanical concentration, lead being of course recovered by the same process. In the roasting of an argentiferous blende there is almost always a considerable loss of silver. The extent of this was recently investigated by K. Sander, who performed his experiments in Malètra and Eichhorn-Liebig furnaces.[2] The ores tested were of the following composition:

Number.	I %	II %	III %	IV %	V %
Zinc	31·00	41·80	43·93	47·50	42·60
Sulphur	22·26	27·40	24·95	33·26	29·07
Lead	7·50	4·00	7·35	2·15	8·75
Iron	4·25	3·00	5·20	13·60	7·50
Lime	1·60	1·40	0·10	.....	0·12
Magnesia	0·65	0·80	0·15	1·20	1·10
Baryta	24·20	10·55	.....	.....	.....
Quartz	2·75	10·20	15·46	1·14	10·00

The results of the roasting were as follows:

Number.	I	II	III	IV	V
Loss %	10·50	10·05	11·25	12·70	12·00
Ag. g. per ton *a*	340·0	413·0	230·0	324·5	375·0
" " " " *b*	335·0	410·0	227·5	330·0	375·0
Ag. loss %	11·77	10·68	12·15	11·22	12·00

a Before roasting. *b* After roasting.

The loss of silver was the larger the higher the temperature maintained.

PULVERIZATION OF THE ORE PRELIMINARY TO ROASTING.—The finer the particles of ore the more rapidly can they be roasted, but the greater the production of dust during the raking on the hearth and consequent loss; and perhaps also the greater the loss of zinc by volatilization if the roasting be done in a reverberatory furnace. Experience has shown that the best size to which to reduce the ore particles is 1 to 2 mm. It is seldom advisable

[1] Bulletin de l'Assoc. Belge des Chim., XVI, 11, 99 to 104.
[2] Zeits. f. angew. Chem., 1902, XV, xv, 335.

to reduce the ore so fine as 1 mm., and 2 mm. is the common size. Mr. Ross, director of the works at Oberhausen, expressed the opinion that blende should not be crushed finer than 2 mm., inasmuch as when the ore is very fine the charge lying on the hearth is not sufficiently permeable to permit the free escape of the sulphurous gas and access of oxygen. He considered that blendes which decrepitate are the easiest to roast, it being unnecessary with them to carry the crushing so far as in other cases.[1]

CONTROL OF THE DEGREE OF DESULPHURIZATION.—The completeness of the desulphurization of blende is of prime importance in the smelting process, since each unit of sulphur combined with zinc in the roasted ore will hold back two units of zinc in the retort when the ore is distilled. It is therefore necessary to maintain a careful watch over the roasting in order to insure the maximum degree of desulphurization that is economical. In general, a tenor of sulphur higher than 1% should not be tolerated in an ore which does not contain calcium or magnesium, but since the degree of desulphurization bears a direct relation to the time and labor of roasting, it is important to systematize the work so that the furnaces will be worked at their maximum capacity corresponding to the desulphurization that is desired. The introduction of mechanical furnaces has largely eliminated the personal equation in so far as output is concerned, and their operation can be governed satisfactorily by assay of their product for the sulphur contained. In hand raked furnaces it has been found a satisfactory system to pay the men according to the quantity of product delivered, a limit being set on the tenor in sulphur. Sometimes the contract price is made on a sliding scale, increasing as the tenor in sulphur falls below the limit and decreasing as it rises above it. If the roasting be done in muffle furnaces, for utilization of the sulphurous gas, it is necessary also to control the strength of the gas in sulphurous anhydride.

There are various rough methods for judging quickly the amount of sulphur still held by the roasted ore. One way is to sprinkle a little of the hot ore from the furnace on some potassium chlorate in an iron spoon, when the presence of unburned sulphide will betray itself by sparking; even the appearance of a few small sparks is considered a sign of good roasting, showing that the sulphur as sulphide has been reduced to about 1%. Another way is to treat a small quantity of the ore in a test tube with chlorhydric acid, the top of the tube being covered with a piece of absorbent paper moistened with a solution of lead acetate. The sulphureted hydrogen which will be evolved will discolor the paper, the degree of discoloration varying according to the percentage of sulphur. A rough quantitative test

[1] Rev. Universelle des Mines, XXV, 47 et seq.

may be made by comparing the color produced in the lead paper with the color produced by a sample of known sulphur contents, tested at the same time and in an identical manner. Since it is inconvenient to weigh out the samples in the roasting house, the quantity of each taken for examination may be equalized approximately by measuring them out in a capsule of suitable size. Of course neither of these tests indicates the presence of sulphur in the form of sulphates, as to which it is always important to know. The experienced roaster can judge closely the progress of the operation towards its end by the evolution of the fumes from the burning ore and their cessation when the oxidation of the sulphur is completed. He will determine the character of the product, within surprisingly narrow limits, by examining it for particles of undecomposed blende, or "shines," but as in the case of the rough chemical determinations a visual examination fails to disclose sulphur remaining in the form of sulphates.

Supply of Air in Roasting Blende.—A furnace employed for roasting zinc blende requires air for two purposes: (1) Combustion of the carbonaceous fuel used for heating the furnace; and (2) oxidation of the sulphur and zinc in the ore. Air is therefore an essential agent in carrying out the process, but frequently there is little attention given either to the quantity or manner of its supply, although the roasting may be delayed by an insufficient supply and the consumption of fuel is increased by an excess. Generally the admission of air to the furnace occurs haphazardly, rather than under scientific control, which in any case would be somewhat difficult with ordinary reverberatory furnaces. In practice, roasting furnaces are a good deal more likely to be run with an excessive, rather than with a deficient, supply of air.

Reverberatory Furnaces.—In reverberatory furnaces the products of combustion draw over the fire bridge, through the roasting chamber, and escape by a flue to the chimney. These gases contain a variable quantity of free oxygen, which has passed through the grate and bed of burning coal, the combustion of coal on an ordinary grate requiring a large excess of air, and has thus become heated, besides a further quantity of oxygen from air which has been introduced cold through the working doors, or slightly heated through ports in the fire bridge, and has been only partially consumed by the ore. With an ordinary grate firing the free oxygen in the gases from the fireplace is likely to be greater in quantity than what is required by even a rich zinc sulphide ore,[1] but because of its dilution with

[1] It is computed in an example, presented further on in this chapter, that 1000 kg. of ore containing 30% S require 1461 cu. m. of air, while in the combustion of 300 kg. of a certain coal with 100% excess of air there will be 2362 cu. m. of unconsumed air passing through the fireplace.

nitrogen and carbonaceous gases its chemical action is weak and if reliance were placed upon it alone, although the theoretically requisite quantity of oxygen were known to be present the roasting would probably proceed slowly and be incomplete in its result. It is necessary therefore to reinforce that oxygen by a fresh supply of air. The more nearly the volume of the latter is kept down to the minimum requisite to produce the maximum chemical activity, i.e., combustion of the ore, and the more it be preheated the greater will be the economy of fuel.

The introduction of the fresh air is sometimes effected through the bridge wall, which is built hollow with ports opening into the roasting chamber, a system which has been found very efficient, although the temperature to which the air admitted in that manner is raised is of course very low. The effect of hot air in roasting is discussed further on in this chapter. Besides such air as is introduced through the bridge wall a good deal is admitted through the side doors, which may be purposely opened and if merely loose plates of iron as in many long reverberatory furnaces always leak more or less anyway. The latter is probably rather advantageous than otherwise, because it causes the admission of the air to be gradual along the length of the hearth, furnishing the supply where it is required directly by the actively burning ore and avoiding the cooling effect of a large volume of cold air sweeping over the nearly spent ore if the air be admitted at the discharge end of the furnace alone. In most reverberatory furnaces, whether Fortschaufelungsöfen or shelf burners, the excess of air over and above what there is in the gases from the fireplace is introduced only through the working doors. When the latter are opened for the purpose of raking the ore there is a great influx of cold air, referred to as false air, which chills the furnace and dilutes the sulphurous gas.

Muffle Furnaces.—In roasting zinc blende in muffle furnaces the products of combustion from the fireplace do not come in contact with the ore, and the oxygen for the oxidation of the latter must be obtained directly from the atmosphere. If the roast reaction takes place according to the proportion

$$ZnS+4O=ZnO+SO_2+O,$$

for every 32 parts of sulphur by weight there must be supplied 64 parts by weight of oxygen to satisfy the requirements of the sulphur and zinc. Therefore in order to burn 1000 lb. of sulphur combined with zinc there are needed 2000 lb. of oxygen, or $2000 \div 0{\cdot}23 = 8{,}695{\cdot}65$ lb. of air. In practice, however, about twice that quantity of air is required, or 17,391·30 lb., which at 62° F. is equivalent to a volume of 17,391·30

×13·141=228,539 cu. ft.; i.e., 1 lb. of sulphur in blende requires 17·391 lb. of air or 228·5 cu. ft. at 62° F.; wherefore 2000 lb. of ore assaying 30% S if roasted to 1% S, liberating 580 lb. of sulphur, requires 10,087 lb. of air =132,553 cu. ft. at 62° F. That quantity of air must be drawn into the roasting chamber by the draught system of the sulphuric acid chambers, or it must be forced in by a blower.

Tenor of Sulphurous Anhydride in the Gas.—In muffle furnaces, used as an adjunct to a sulphuric acid plant, it is necessary to control closely the admission of air to the roasting chambers, not only to avoid the cooling effect of an excess of cold air (which may be obviated to a large extent by preheating the air), but also to avoid diluting the sulphurous gas below 4% SO_2 by volume, which is about the minimum limit for successful acid manufacture; it is objective to produce as strong a gas as possible. If the roasting reaction take place according to the proportion cited above, which may be transposed to

$$2ZnS+4 \text{ vols. } (O_2)+4\times\tfrac{79}{21} \text{ vols. } (N_2) = 2ZnO+2 \text{ vols. } SO_2+1 \text{ vol. } (O_2)+4\times\tfrac{79}{21} \text{ vols. } (N_2),$$

the theoretical composition of the roast gas by volume is 11·08% SO_2, 5·54% oxygen and 83·38% nitrogen.[1] If twice that quantity of air be used, as is common in such roasting, the proportion will be

$$2ZnS+8 \text{ vols. } (O_2)+8\times\tfrac{79}{21} \text{ vols. } (N_2) = 2ZnO+2 \text{ vols. } SO_2+5 \text{ vols. } (O_2)+8\times\tfrac{79}{21} \text{ vols. } (N_2),$$

and the tenor of SO_2 in the roast gas will be 5·39% by volume, which is approximately what is obtained ordinarily in practice, although some well designed and carefully managed furnaces afford a gas with as high as 7% SO_2, and even more.

It is frequently stated that blende is inferior to pyrites in the manufacture of sulphuric acid, because its gas is less strong in sulphurous anhydride. However, although pure blende contains only 32·85% S, against 53·33% S in pure pyrite, the difference in the SO_2 tenor of their roast gases is insignificant. In this connection R. Hasenclever presented the following computation of the strength of the gas that can be obtained from sulphur, pyrites and brimstone.[2] One hundred kilograms of sulphur burned to sul-

[1] One volume of sulphur in combining with two volumes of oxygen produces only two volumes of sulphurous anhydride; similarly in combining with three volumes of oxygen there are only two volumes of sulphuric anhydride produced.

[2] Chem. Ind., 1884, pp. 78 to 86; Journ. Soc. Chem. Ind., 1884, p. 630.

phurous acid require 69·8 cu. m. of oxygen, and yield 69·8 cu. m. of sulphurous acid. Mixed with the oxygen there are 262·2 cu. m. of nitrogen. To convert 69·8 cu. m. of SO_2 into SO_3 there are required further 34·9 cu. m. of oxygen, which are mixed with 131·1 cu. m. of nitrogen. Inasmuch as the gases should contain 6% of free oxygen at the escape from the chambers, it is necessary to introduce 143·8 cu. m. of air in excess. The gases from 100 kg. of sulphur will therefore amount to 69·8+262·2+34·9 +131·1+143·8=641·8 cu. m., and their composition by volume will be 10·64% SO_2, 10·36% O, and 79% N. A similar calculation for 100 kg. of pyrite and blende gives the following results:

100 kg. pyrites: 37·2+52·3+140+18·6+70+104·9=423 cu. m., consisting of 8·79% SO_2, 9·59% O and 81·62% N.

100 kg. blende: 23+68·5+11·5+43·2+11·5+43·2+75·8=276·7 cu. m., consisting of 8·31% SO_2, 14·05% O and 77·64% N.

The gas mixture from 100 kg. consists therefore of

Constituents.	Sulphur.	Pyrites.	Blende.
Cu. meters SO_2.....	69·8	37·2	23·0
" " N.....	262·2	192·3	129·7
" " Air.....	309·8	193·5	124 0
" " Total....	641·8	423·0	276·7
% SO_2 by volume...	10·6	8·8	8·3

However, the total volume of gas per 100 kg. of real sulphur in brimstone, pyrites and blende is respectively 612, 800 and 840 cu. m., wherefore although the gas obtained by burning blende may be almost as rich as that from pyrites the chamber space must be increased in the ratio 800 : 840.

HEAT REQUIREMENTS IN BLENDE ROASTING AND USE OF PREHEATED AIR. —The oxidation of zinc sulphide is exothermic, i.e., it develops heat and theoretically does not require the application of extraneous heat, as is necessary in the reduction of zinc oxide, which is an endothermic reaction. The heat of combination or decomposition of ZnS is, according to Naumann, 41,880 gram-calories, that of ZnO, acording to the same authority, 85,430;[1] the oxidation $S+2O=SO_2$ evolves 71,000 gram-calories. Consequently the reaction

$$ZnS+3O=ZnO+SO_2$$
$$41{,}880 \qquad 85{,}430+71{,}000=+114{,}550$$

[1] This is also the figure reported by Thomsen. Roberts-Austen, in Introduction to the Study of Metallurgy, gives the heat of formation of ZnS in round numbers 43,000 gram-calories; of ZnO, 86,400; of SO_2, gaseous, 71,000.

evolves a large quantity of heat. The reactions which take place practically in blende roasting are a good deal more complicated, involving the formation of sulphates and their subsequent decomposition, the former being an exothermic and the latter an endothermic process, while there is always present an excess of oxygen and its diluting nitrogen, which absorb heat.

Consequently in roasting zinc blende there is the necessity first to raise the temperature of the ore and air to the point where ignition begins, after which there is an evolution of heat sufficient to maintain the requisite temperature in many furnaces, even in grate kilns, until the sulphur is burned down to 8% (in Gerstenhöfer furnaces, roasting fines, down to 5%) beyond which point the heat generated by the sulphur alone is insufficient to make up for the loss by radiation from the furnace and absorption by the cold air passing over the ore, and the temperature falls below the ignition point of the zinc sulphide; so combustion ceases. In order to complete the roast therefore it is necessary toward the end to supply heat from the combustion of carbon, which is also required to break up the zinc sulphate that has previously been formed. These theoretical considerations are borne out in practice, wherein it has been found that:

1. The decomposition of zinc sulphate requires a high temperature at the end of the roast.

2. The oxidation of zinc sulphide develops sufficient heat to proceed independently until the sulphur has been burned down to a percentage varying according to the furnace and the conditions.

3. As a consequence of the second conclusion, the length of a furnace with a single fireplace at one end is dependent upon the tenor of sulphur in the ore.

4. In a muffle furnace, wherein the flames are carried in a flue under each hearth in the opposite direction to the travel of the ore, there may be a point where the temperature in the flue is lower than that in the hearth chamber and the gases of combustion passing through the flue may actually have a cooling effect instead of a heating.

The last conclusion is based on the experience in Europe with muffle furnaces of the new Hasenclever type, in which the middle flue has now been generally abandoned, the flames being carried under the lowest hearth and back over the uppermost, serving to raise the ore on the latter more quickly to the temperature of ignition.

Relation Between Air Supply and Consumption of Fuel.—In a previous section of this chapter it has been shown that if blende be roasted with eight parts of oxygen, yielding a gas containing 5·39 SO_2 by volume (which would be a fair practical result), 2000 lb. of ore assaying 30% S and liberating

580 lb. of sulphur require, 10,087 lb. =132,553 cu. ft. of air. This weight and volume of air must all be raised to the temperature of the furnace, say 600° C. as the temperature of the coolest hearth, in order to permit the roasting to proceed. As a matter of fact, experience shows that the heat generated by the sulphur so long as the latter is present in quantity exceeding 8% of the weight of the ore is sufficient to heat the quantity of air corresponding to a percentage of 5·39 SO_2 by volume in the roast gas, and the gas is sufficiently hot for the successful operation of a Glover tower. At the end of the roast, however, it may be assumed that a temperature of 900° C. must be attained in the furnace, and there being then little or no sulphur in the ore available as fuel the requisite heat is attainable only at the expense of the coal burned, the heat of which in the case of a muffle furnace must be abstracted by conduction through the tile hearth of the muffle. It is obvious that the greater the volume of air admitted to the furnace the greater is the quantity of heat required to raise it to the temperature of the furnace.

Method of Admitting the Air.—The quantity of fuel required depends not merely upon the volume of air admitted into the furnace, but also upon the manner in which it is introduced. It is manifest that more air is required in those parts of the furnace where ore rich in sulphur is burning than toward the discharge end where the tenor in sulphur has been reduced to 8% and less, part of which is then existing in the form of sulphate. A high temperature and comparatively little oxygen are required there; an excess of air has the effect of cooling the ore below the temperature of combustion of its remaining sulphides and dissociation of its sulphates, and the process will be checked unless the necessary heat be supplied extraneously, leading to a waste of fuel, which may be a large one.

This may be illustrated by supposing that all the air required by the ore were admitted at the discharge end of the furnace, sweeping over the nearly burned out ore. The temperature that has to be maintained there is approximately 900° C., in order to insure decomposition of the sulphates. The air passing through must be raised to that temperature or the ore will be cooled below it. Ignoring the small quantity of heat which the ore itself is still capable of furnishing, the air must be heated by the combustion of coal. A simple physical calculation shows how much must be consumed for that purpose.

In order to heat 1 kg. of water from 15° to 900° C., there is required 885 kg.-calories. The specific heat of air under constant pressure being 0·2375, the quantity of heat required to raise 1 kg. of air from 15° C. to 900° C. is $885 \times 0{\cdot}2375 = 210{\cdot}1875$ calories. In order to heat 10,087 lb.

=4575 kg. from 15° to 900° C. there would be required 210×4575= 960,750 kg.-calories. If coal of a calorific power of 8000 kg.-calories be assumed, the heating of 4575 kg. of air would require 120 kg. of coal if the entire efficiency of the coal were realized. If only 50% of the calorific power of the coal were realized there would be required 240 kg., i.e., about 26·5% of the weight of the ore (2000 lb.=907 kg.). The admission of only one half that volume of air would reduce the consumption of coal to 13·25%. It is assumed that the air is admitted at atmospheric temperature.

Practically the roasting of zinc blende can be done with the consumption of 15 to 20% of the weight of the ore in good coal, but careful management of the fire and proper admission and regulation of the air supply are necessary to obtain such a result. On the other hand poor coal, combined with poor management and poorly designed furnaces, may lead to results so unsatisfactory as one ton of coal per ton of ore. Muffle furnaces, especially when gas fired, show as high economy in fuel as reverberatory furnaces, because of the more perfect control of the air supply that is possible with them. For the same reason mechanically raked furnaces are likely to show improved results.

Advantage of Preheating Air.—The advantage that can be gained from preheating the air of oxidation for blende roasting has been recognized by many metallurgists and numerous furnaces have been designed with that in view, but although uniformly good results have been apparently obtained, both as to economy of fuel and hastening of the calcination, there have been few furnaces designed and constructed so as to give the maximum efficiency. The muffle furnaces of Hasenclever and Eichhorn & Liebig, which are so largely used in Germany at the present time, are generally arranged with ducts under the combustion flues through which the air for oxidation of the blende and combustion of the coal is drawn and becomes more or less heated in its passage. Spence furnaces have been built for roasting pyrite with air ducts in the side walls. There have been mechanical furnaces in which the stirring arms have been cooled by circulation of air through them, the hot air being returned to the furnace. While Blake in designing his circular furnace (q. v.) which was used in Wisconsin, went so far as to preheat the air for the roasting chamber by passing it through brick checker-works heated alternately by an independent fireplace. Each of those methods of air heating has given good results, although no one of them is a recuperative system or accomplishes the maximum economy which can be expected from an application of the latest principles of furnace design. In this respect the practice in roasting furnace construction is behind that of the distillation and other kinds of heating furnaces.

The rational design of a furnace for blende roasting (the principles apply equally well to most types of heating furnaces) is in the first place to build such massive brickwork as will most completely store and retain the heat generated in the furnace; in the second place to abstract in the furnace itself, or in an accessory recuperator, so much of the caloric that the escaping gases will be no hotter than is necessary to produce the requisite chimney draught. If the draught were to be produced mechanically the escaping gas could be cooled to a very low temperature indeed.

The heating of air in ducts in the side walls obviously takes heat from the active part of the furnace; the radiation from the walls may be diminished to a certain extent, but practically the only useful agency of such a system is to transfer heat from one part of the furnace, where perhaps it is not needed, to another part where it is desirable. The circulation of air through the stirring mechanism is also directly cooling in its action; the system is especially useful on the ground of preservation of the stirring mechanism, but from the view point of heat recuperation it can accomplish nothing beyond a transferal of what is still effective heat. Similarly the installation of air ducts under the combustion flues is essentially a system of transferring heat that is not yet waste heat from one place to another, save for what is gained by cutting off conduction into the foundation of the furnace, which must naturally be small; it is obvious that the circulation of a current of cold air under the flues, which are designed primarily for the generation of heat, can obtain heat only from those flues, which must of necessity be cooled in the process; if the flues have more heat than is needed, which is probable at certain places in their course, something may be gained by robbing them there and carrying the heat back to the discharge end of the furnace, where the sulphur in the ore is lowest and most heat is required.

The general experience in favor of preheating air by the above methods indicates that a mere transferal of heat may be useful. Blake's system of preheating the air by independent fireplaces, on the other hand, actually adds to the caloric of the furnace, but since it is obtained at the expense of a direct combustion of coal it cannot possibly lead to any economy of fuel (and probably involves an actual loss), so that any advantage ascribed to it must be due rather to the more intense chemical activity of hot air, offsetting an increased consumption of fuel.

The most effective method of increasing the efficiency of a roasting furnace is by a recovery of heat from the products of combustion after they have escaped from the furnace in which the maximum direct effect has already been obtained from them. Whatever heat in the form of air can

be thus restored to the furnace is an actual gain by recovery and must logically lead to economy in fuel. The means for effecting such a recovery are discussed in the chapter on "Fuel and Systems of Combustion." It may be remarked here, however, that the recovery is limited to the surplus of heat in the waste gases over what is required to give the requisite draught in the chimney, though we may see the time when chimneys are dispensed with and the draught is produced by mechanical exhausters after the gases have been robbed of nearly all their heat, in which direction American steam boiler practice is now tending.

We may say therefore that if a roasting furnace be built of such massiveness as to reduce to the minimum loss of heat by radiation, if the heat of the sulphurous gas is utilized to the maximum in the Glover tower, and if the maximum quantity of heat above what is required by the chimney is recovered and restored to the furnace, the highest efficiency of fuel will have been obtained that is to be expected under present conditions of practice.[1]

PROGRESSIVE DESULPHURIZATION.—In roasting zinc blende the desulphurization proceeds gradually from the feed end to the discharge end of the furnace. Samples of the partially burned ore at intermediate points should show a steady diminution in the percentage of zinc present as sulphide, and an increase in the percentage of zinc oxide. The percentage of zinc sulphate will first increase and then decrease. The progressive desulphurization is shown by the subjoined analyses.

I. An Eichhorn-Liebig furnace, with six muffles, roasting 3000 kg. of ore per 24 hours, gave the following results:

Raw ore.	27·8% S	31·2% S	31·2% S	31·2%S	31·2%S
End of 1st hearth.	24·9	28·0	23·8	24·3	24·2
" " 2nd " .	17·3	24·8	22·7	19·7	21·5
" " 3rd " .	13·2	16·1	16·5	12·3	17·3
" " 4th " .	2·3	8·8	12·5	9·9	.....
" " 5th " .	0·2	7·8	7·8	5·4	5·6
" " 6th " .	0·1	0·96	0·9	1·29	1·33

A charge was drawn every six hours, having been 36 hours in making the passage through the furnace.

II. An Eichhorn-Liebig furnace, with four muffles, it having been

[1] The opinion as to the advantage there may be in a mere transferal of heat by air from one part of a furnace to another is confirmed by the results obtained with reverberatory copper smelting furnaces, in which the air is preheated in that manner. It would be interesting to know precisely how much better results might be obtained both in blende roasting and copper smelting by preheating the air in a way which does not necessarily abstract heat from the active part of the furnace.

found unnecessary to provide more than that number, gave the following results in roasting Silesian blende:

Raw ore.	19·2% S	26·8% S	26·8% S	26·5% S	26·5% S
End of 1st hearth.	17·6	19·10	21·9	15·4	21·3
" " 2nd " .	12·0	11·20	14·3	9·9	12·4
" " 3rd " .	3·4	1·02	1·48	0·75	1·06
" " 4th " .	0·6	0·35	1·02	0·75	1·06

The charges in this furnace were introduced at intervals of eight hours and remained in the furnace 32 hours. The furnace, which was gas-fired, roasted 3500 to 4000 kg. of ore (yielding 3000 to 3500 kg. of product) with the consumption of 980 kg. of coal (containing 12 to 16% ash). The air supply was preheated in channels in the furnace. The roast-gas had a tenor of 5 to 6% SO_2 and entered the dust chamber at a temperature of about 400° C. The furnace crew comprised two men per 12-hour shift.[1]

A description and engraving of the Eichhorn-Liebig furnace will be found on pages 129 and 130 of the next chapter.

III. Samples from a simple reverberatory furnace at the Austro-Belge works at Corphalie, Belgium, gave the following results:[2]

Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide	Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide
Charged	64·5%	0%	0%	4	17·5%	8·4%	39·0%
2	58·0	3·4	4·6	5	10·0	2·6	50·5
3	38·0	7·9	19·0	Drawn	1·2	2·2	59·7

IV. A reverberatory furnace with two hearths at the Vieille Montagne works at Flône, Belgium, showed:[2]

Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide	Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide
Charged	83·0%	0%	0%	6	23·2%	12·3%	57·8%
2	70·5	3·7	15·2	7	17·7	7·8	65·6
3	52·2	3·9	34·6	8	8·6	6·2	75·5
4	51·5	4·2	38·0	Drawn	1·9	5·9	81·0
5	43·0	11·0	41·5				

The furnaces employed at Flône have hearths of 6·5×2 m. and roast 2850 kg. of ore per 24 hours with consumption of 500 kg. of coal (vide p. 77).

[1] K. Eilers, Trans. Am. Inst. of Min. Eng., XX, 336 et seq.
[2] Létrange, Réduction de Minerais de Zinc par Electricité, p. 10.

V. The following determinations were from a Hasenclever furnace at Stolberg, Germany:[1]

Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide	Sample	Zinc Sulphide	Zinc Sulphate	Zinc Oxide
Charged	57·2%	0%	0%	9	24·2%	5·2%	30·0%
2	53·5	3·5	0·8	10	19·2	5·8	32·6
3	53·0	3·8	3·4	11	10·0	7·8	40·0
4	48·4	4·2	4·6	12	7·9	6·2	44·9
5	39·9	4·3	12·3	13	1·5	4·7	52·5
6	35·2	5·0	16·5	14	1·6	2·6	53·0
7	34·8	6·9	17·0	Drawn	1·2	0	55·0
8	25·2	6·3	27·1				

TEMPERATURES ATTAINED IN BLENDE ROASTING.—Experience has shown that a temperature of approximately 900° C, corresponding to orange color, i.e., a color intermediate between light cherry red and full yellow, must be attained in the blende roasting furnace to insure the requisite desulphurization. The heat is furnished partly by the burning ore itself and partly by the combustion of coal, the latter being required at the end of the roast when the ore has nearly burned out. The reaction $ZnS+3O=ZnO+SO_2$ corresponds to the evolution of 114,550 calories of heat, i.e., 1 kg. develops $114,550\div97=1181$ calories. An ore containing 90% blende will therefore develop $907\times0.9\times1181=964,050$ calories per 907 kg. (2000 lb.) burned. According to John Hollway[2] the temperature of combustion in air of zinc sulphide is 1992° C.; of ferrous monosulphide, 2225° C.; of pyrites, 1719° C. Carbon burned to CO_2, without excess of air, affords a temperature of approximately 2680° C. In all cases the temperatures of combustion are reduced by the excess of air which is practically required. Thus if carbon be burned with 100% excess of air, which is not uncommon, the temperature of the fire is reduced to approximately 1390° C., even if it all be burned to CO_2. It is evident therefore that in a furnace of only moderate length there will be a point where the temperature generated by the burning ore may be higher than that of the gases from the fireplace, which is found practically to be the case in many furnaces.

The actual pyrometric measurements of blende roasting furnaces in operation have not been extensive. Horace F. Brown has reported some data concerning the temperature attained in a horseshoe furnace at Collins-

[1] L'étrange, loc. cit.

[2] In a paper entitled "A New Application of Bessemer's Process of Rapid Oxidation, by Which Sulphides are Utilized for Fuel," read before the Society of Arts (London), February 12, 1879. Previous to Hollway, Bode in his treatise on "Sulphuric Acid" gave the following temperatures of combustion: FeS_2, 2586° C.; FeS, 2725° C.; ZnS, 2850° C.

ville, Ill.[1] This furnace had a hearth of 135 ft. effective length, with four fire boxes at intervals in that length. The ore roasted assayed about 58% Zn and 29 to 30% S. The temperature from the feed end to the first fire box, a distance of 35 ft., was found to average 648° C. (1200° F.); from the first fire box to the second, 35 ft., 815° C. (1500° F.); from the second to the third, 955° C. (1750° F.); and from the third to the fourth, 1093° C. (2000 F.). At the first fire box the sulphur tenor of the ore had been reduced to 18 or 20%; at the second to 10 or 12%; at the third to 5·5 or 6%; and at the fourth (discharge end) to 0·8%. The ore bed was 2 in. deep. It was moved forward at the rate of 8 ft. per hour, remaining in the furnace 17 to 18 hours.

Pyrometric measurements at Oberhausen with the new Hasenclever (Rhenania) furnace showed a temperature of 580° to 690° C. in the first muffle and 750° to 900° C. in the second and third. With ore rich in sulphur the middle muffle was the hottest, but with ore poor in sulphur the highest temperature was registered in the lowest. In the Hasenclever furnaces at Stolberg a temperature of 1000° C. is sometimes attained.

The sulphurous gas from a muffle furnace will escape at a temperature of approximately 500° to 600° C. Its temperature upon reaching the Glover tower will depend of course upon the velocity and distance it has to travel, and the precautions taken to conserve heat during the passage. It is important to settle the dust as completely as possible and at the same time avoid loss of heat by radiation, which implies that the dust chambers and flues should have thick, non-conducting walls. It is desirable to deliver the gas to the Glover tower as hot as possible—up to about 650° C.—temperatures of 320° C. to 430° C. being common in Glover tower operation, in which case the gas will leave the tower at 88° to 93° C. The Glover tower is a true heat recuperator; the hotter the gas delivered to it, the greater the economy in the combined process of blende roasting and sulphuric acid manufacture.

Volume of Gaseous Products.—It has been computed, in a previous section of this chapter, that if blende be roasted in such a way as to produce a gas containing 5·39% SO_2 by volume, which is a fair practical result, there is used eight parts by weight of oxygen per part of blende instead of three parts as required by theory, wherefore 2000 lb. of ore containing 600 lb. of sulphur, of which 580 lb. are burned off, must be supplied with 10,087 lb. of air =132,552 cu. ft. at 62° F. (16·7° C.). The oxygen of that quantity of air is partly taken up in combination by the zinc and sulphur, but chiefly passes through the furnace unaltered, diluting the gas of com-

[1] In a private communication.

bustion. The 580 lb. of sulphur burned require 580 lb. of oxygen and produce 1160 lb. of sulphurous anhydride; the 1160 lb. of zinc require 290 lb. of oxygen.[1] Consequently 580+290=870 lb. of oxygen have been fixed, and the products of combustion are then 10,087 lb. air—870 lb. oxygen +1160 lb. sulphurous anhydride =10,377 lb. The original 10,087 lb. of air contained 2320 lb. of oxygen and 7767 lb. of nitrogen. The products of combustion must consist therefore of:

By Weight.	By Volume.
7767 lb. N — 75% 1450 lb. O — 14% 1160 lb. SO_2 — 11%	99183 cu. ft. — 81·4% 16252 cu. ft. — 13·3% 6504 cu. ft. — 5·3%
10377 lb. 100%	121939 cu. ft. — 100·0%

Résumé: 2000 lb. of blende containing 1200 lb. (60%) zinc and 600 lb. (30%) sulphur, of which 580 lb. are burned off, requires to produce a gas suitable for the manufacture of sulphuric acid the supply in round numbers, of 10,000 lb. of air, or 132,500 cu. ft. at ordinary temperature; and yields 122,000 cu. ft. of sulphurous gas reduced to 0° C., the actual volume being proportional to the absolute temperature at which it escapes from the furnace.

In a muffle furnace the gases arising from the combustion of the coal are of interest only in so far as they enter into the calculation of heat recuperation and the means of causing the draught. In a reverberatory furnace on the other hand all the gases go off together, and their combined volume must be considered in planning the dimensions of flues, dust chambers, etc. The calculation becomes then more complicated and is less certain, because the excess of air required and the false air admitted when the ore is being raked are rather variable factors.

Assuming that the coal to be used contains 75% carbon, 5% hydrogen, 5% oxygen, 1% nitrogen, 1% combustible sulphur (i.e., sulphur not already combined with oxygen) and 10% ash, the combustion of 1000 kg. of such coal requires air and yields gaseous products as follows:[2]

[1] For convenience in calculation one part of sulphur is assumed to require two parts of zinc, and one part of oxygen four parts of zinc, by weight.

[2] For sake of simplicity in the calculation the coal is assumed to be free from moisture: similarly the small percentage of moisture in the air used for its combustion is disregarded. This hypothetical coal corresponds approximately to the composition of Pittsburgh, Penn., bituminous, which has a calorific value of about 7500 calories, or about 8300 when dry and free from ash. The presence of 8% oxygen with 5% hydrogen leaves 4% of the latter available.

Kg. of Combustible.	Burned to	Requiring cu. m. of Air.	Producing cu. m. of Gas. *a*
750 C.....	CO_2	6765	6743
40 H.....	H_2O	1074	1299
10 H, 80 O.....	H_2O	.....	325
10 N.....	.....	.....	8
10 S.....	SO_2	33	34
Total.....	.....	7872	8409

a Direct products of combustion plus nitrogen of the air used.

If 100% excess of air be used in the combustion of the coal the total quantity required is $2\times7872=15{,}744$ cu. m., and the volume of the gases of combustion is $8409+7872=16{,}281$ cu. m., ignoring moisture.

In burning 1000 kg. of ore assaying 60% Zn and 30% S, 29 units of the sulphur being consumed, there is required 290 kg. of oxygen for the sulphur and 145 kg. for the zinc, a total of 435 kg. =1890 kg. (1461 cu. m.) of air. The product of the combustion will consist of 580 kg. (203 cu. m.) of SO_2, plus 1455 kg. (1160 cu. m.) of nitrogen, a total of $203+1160=1363$ cu. m. of gas,[1] no excess of air being yet reckoned. If now the roasting be effected with a consumption of 30% coal, for every 1000 kg. of ore calcined 300 kg. of coal will be burned. The quantity of false and excess air that will be introduced into the furnace can only be guessed at; we may say 10 times the quantity actually required by the ore, i.e., $10\times1461=14{,}610$ cu. m.; the percentage of SO_2 in the gas escaping from the furnace will be to a certain extent a check on this estimate. The combined gases escaping from the furnace will be consequently as follows:

From ore	1,160	cu. m.	containing	203	cu. m. SO_2
" coal.....	4,884	" "	"	2	" " "
Excess air.....	14,610	" "			
Total gas	20,654	" "	"	205	" " "

The gases of combustion estimated above contain a trifle less than 1% SO_2 by volume, a result which corresponds with analyses of the roast gases of many furnaces. In general the gases of reverberatory furnaces contain from 1 to 2% SO_2. A tenor of 2% would correspond approximately to the use of false and excess air to the extent of four times the quantity actually required by the ore, all conditions remaining as specified above.

[1] $203\div1363=14{\cdot}9\%$ is the maximum percentage of SO_2 by volume that is theoretically possible in roasting blende in air, with no excess of the latter.

The weights and volumes used above are reduced to the equivalents of tons of 2000 lb. by multiplying by 0·907; cubic meters are converted to cubic feet by multiplying by 35·314.

Résumé: 2000 lb. of blende containing 1200 lb. (60%) zinc and 600 lb. (30%) sulphur, of which 580 lb. are burned off, if roasted with 30% of a superior coal and with such excess of air that the escaping gases will contain about 2% SO_2 by volume, requires for both ore and coal the supply of 385,000 cu. ft. of air, in round numbers, and produces 340,000 cu. ft. of sulphurous and carbonaceous gas, that volume being reduced to 0° C and the actual volume being proportional to the absolute temperature of the escaping gas.

The quantity of air used and the quantity of sulphurous gas produced are therefore nearly three times as great in roasting in a reverberatory furnace as in roasting in a muffle furnace. If the air used in the fireplace of the muffle furnace and the gases escaping therefrom were included in the estimate of that type the quantity of air required and the aggregate volume of gases produced would be more nearly the same in each case, depending largely upon the proportion of excess air employed in the combined operation.

Muffle Furnaces Compared with Reverberatory Furnaces.—The chief difference in principle between roasting in a muffle furnace and roasting in a reverberatory has been pointed out in the foregoing discussion of air supply and heat requirements. In the former the gases of combustion pass directly over the hearth, an excess of air in said gases being necessary, while in the latter the gases do not come in contact with the ore and the air is admitted to the hearth chamber independently, wherefore it is possible to regulate the supply of air more exactly. So long as the heat generated by the burning sulphur is greater than that furnished by the combustion of fuel, the ore should roast on the hearth of the muffle as rapidly as on the hearth of the reverberatory, frequency of stirring and other conditions being equal. But when the aid of extraneous heat is required as at the end of the roast especially, and to a less extent at the beginning, when the ore must be heated to the temperature of ignition, the muffle furnace is at a disadvantage, because heat is available neither by contact of the hot combustion gas or by luminous radiation, but must be transmitted through 2 to 4 in. of fire clay tiling, which is comparatively a poor conductor of heat.

The comparatively high non-conductivity of the tile of the hearth in a muffle furnace involves the necessity of a high temperature in the combustion flue beneath it in order to transmit heat rapidly enough to make good what is taken away by the air passing over the hearth. If the balance be

not maintained the hearth will be cooled. Of course, it follows that the thinner the hearth is, the more rapidly is the heat transmitted and the more easily is the temperature maintained in the roasting chamber. For structural reasons, however, it is hardly feasible to construct the hearth of tiling less than 2 in. thick. Theoretically it would appear that, with the same temperature and quantity of heat developed from the fireplace, the muffle furnace should be slower than the reverberatory; that in order to accomplish the same work there should be needed a greater hearth area and a larger consumption of coal. Practically, the muffle furnaces appear to roast as much ore per square foot of hearth and per man as the reverberatories, and on the whole show a smaller consumption of coal per ton of ore.

MECHANICALLY RAKED FURNACES COMPARED WITH HAND RAKED.—The use of mechanically raked furnaces, muffle or reverberatory, is of comparatively recent introduction in the metallurgy of zinc. Save for the Hegeler furnace, which was built at Lasalle, Ill., in 1881, where it has been in successful operation since that time, and the turret furnaces which were introduced at Oberhausen, in Westphalia, about 1884, there was practically no mechanical blende roasting done until a Brown horseshoe furnace was built at Collinsville, Ill., about 1890. Since that time mechanical furnaces have been generally adopted in the United States, where their work has been so satisfactory that now no zinc smelter would think of reverting to hand raked furnaces.[1] In Europe the mechanical furnaces have not yet found much favor and the roasting is still done chiefly by hand, although mechanical furnaces were used there experimentally and temporarily even before the Hegeler furnace was installed at Lasalle.[2] European zinc smelters have been prejudiced against mechanical furnaces, however, partly because they have feared that the requisite degree of desulphurization could not be attained and partly because the comparative cheapness of labor in Europe has not offered a strong incentive to its economy. Both these objections have been refuted by the American experience, which has shown that the desulphurization can be carried with a mechanical furnace as low as desirable, while the cost of roasting is a good deal lower for labor in the United States than in Europe, notwithstanding that wages are a good deal higher.

The only difference, theoretically, between roasting manually and me-

[1] This question has been greatly complicated by the recent litigation with respect to certain of the successful types of furnaces, which has restricted their use.

[2] The mechanical furnace invented by Kuschel and Hinterhuber, which was a reverberatory with a circular revolving hearth and fixed rakes, was described in Kärnthn. Ztschr., 1871, p. 169, and Berg- u. Hüttenm. Ztg., 1871, p. 320, and 1872, p. 200. This furnace was used for blende roasting at the Johannisthalerhütte in Unterkrain, Austria. The Brunton furnace (circular, revolving hearth) and Parkes furnace (circular, stationary hearth), used for copper ore, antedate 1860.

chanically is that in the latter way the ore can be stirred much more frequently and more regularly than by hand with the same number of men, which is equivalent to saying that the same result can be obtained with fewer men. This difference is so great that two men per 12 hours or four men per 24 hours can attend to a furnace that will roast 20 tons of raw ore per day, while to accomplish the same result with hand raked furnaces a force of 12 to 16 men per 24 hours would be needed under the ordinary conditions of practice. It is true that a higher class of labor is required on the mechanical furnaces than on the hand raked, but after making allowance for that the advantage is greatly in favor of the former. Thus if four men, each paid $2·50 per day, can roast 20 tons of ore the cost for labor is only 50c. per ton; while with 16 men @ $1·75, roasting the same quantity the cost is $1·40. The difference is partially offset, however, by the power which must be supplied to the mechanical furnace and the increased expense for repairs and renewals, besides the greater first cost.

An important advantage of the mechanical furnaces which it is difficult to reckon in dollars and cents is that the personal factor is largely eliminated. The physical equations as to air supply and heat requirements are the same for the mechanical and the manual furnaces, but because of the more frequent raking the oxidation takes place rapidly in the mechanical furnaces and there is consequently a high degree of hearth activity.

The objection is sometimes raised to mechanical furnaces that as the rakes pass under the ore feed a certain quantity of raw ore may lodge on them and be carried in advance of its normal progress before it falls off, thus leading to the presence of imperfectly roasted particles in the product discharged from the furnace. This objection is easily overcome, however, by shutting off the ore feed when the rake is passing under the chute, as is done in most well designed mechanical furnaces.

The application of mechanical raking does not lend itself so easily to muffle furnaces as to reverberatories, because of the necessity of closely regulating the air supply of the former. The excellent furnace of Ropp could hardly be built with a muffle in any way. On the other hand, the Hegeler furnace is an example of a successful mechanically-raked muffle furnace for blende roasting; also the Falding furnace; while both the Spence and Herreshoff, which are used successfully for roasting pyrites, can be undoubtedly adapted to blende roasting, and the Pearce turret and Brown straight line furnaces both have been built with muffles.

Methods of Heating Roasting Furnaces.—The extraneous heat required by a roasting furnace may be supplied from an ordinary fireplace with a grate, by firing with natural gas or producer gas, or by the waste

heat of another furnace. Of these methods, direct firing from a grate is by far the most common, while the employment of waste heat is seldom practised. The principles of the combustion of fuel are discussed in a subsequent chapter and reference should be made thereto, since they are the same for roasting furnaces as for distillation or other heating furnaces.

Direct Firing.—In direct firing, flat grates, step grates and clinker grates are used, according to the character of the fuel; the ash pit may be entirely open, or it may be provided with a door by which the admission of air can be regulated, or it may be operated with a closed door and a blast of air under the grate. In Kansas the shelf burners and two hearth reverberatories have generally open ash pits and a few wrought iron bars for a grate, on which there is carried a bed of clinker and a thick bed of coal, the draught being regulated by the damper in the chimney. At the Empire works, at Joplin, step grates with an under-grate blast were used. In Europe both flat grates and step grates are to be found. The arrangement of the ash pit with doors which can be closed is preferable to open ash pits, because of the control that can be had on the admission of air, even if the air be not blown in under pressure, when of course the ash pit door must be closed tightly.

In a hand raked reverberatory furnace there is usually only one fireplace at the discharge end of the furnace; the shelf kilns used in Kansas are arranged in such a way that one fireplace serves two furnaces. With the long mechanically raked reverberatory furnaces there are necessarily several fireplaces, disposed at intervals along the sides (or in the case of circular furnaces, around the periphery) because the effective heat of a single fireplace at the discharge end would fall far short of the zone of active combustion of the ore itself. A fireplace at the end, moreover, would interfere with the movement of the raking mechanism of most of these furnaces. The arrangement of the fireplaces of the mechanical furnaces is shown in the drawings of the Brown, Pearce, Ropp and others, which are described in the next chapter.

Natural Gas Firing.—In firing the long furnaces with natural gas in Kansas small pipes discharge gas into the roasting chamber through openings at intervals in the side walls. The gas pipes may or may not terminate with burners to mix air with the gas, on the principle of the Bunsen burner. The gas being commonly, though erroneously, considered to be costless, or at all events as something which is very cheap, is burned without much regard to economy and a volume of cold air is often admitted to the roasting chamber which is limited only by the draught of the chimney and the size of the ports. In the shelf kilns used in Kansas the gas is com-

monly introduced by a burner on each side of the door of the lowest hearth, additional air for the ore being admitted through a register in the door.

Producer Gas Firing.—In firing a roasting furnace with producer gas, the producer may be attached directly to the furnace or it may be situated at a distance of many hundred feet away, though for economy in fuel it is best placed near the furnace, if not directly connected with it. This method of firing is rarely employed with other than muffle furnaces. The gas is led to the combustion flues under the muffles, where the secondary air is introduced, either in a mixing chamber from which the products of combustion pass into the flues, or directly into the flues at intervals of their length, or in both ways. By introducing the air at intervals in the length of the flues the flames can be elongated so far as any combustible gas remains unconsumed, and the method being analogous to placing fireplaces at intervals on the sides of a long reverberatory furnace the muffle hearths may be extended to great length, as in the case of the Hegeler furnace at Lasalle, Ill. Lasalle and Peru, Ill., are the only places in the United States where producer gas firing is employed in blende roasting; in Europe its use is more common.

Use of the Waste Heat from Distillation Furnaces.—In Germany and Wales roasting furnaces were formerly connected with the distillation furnaces and heated by the flames escaping from the latter, but if there are any furnaces arranged in that way at present outside of Wales, I am unaware of it. The practice still exists, however, in the United States at Nevada, Mo., and Wenona, Ill., where shelf burners are set above the distillation furnaces. The distillation furnaces have single combustion chambers, i.e., only one brigade of retorts, and face in opposite directions with a distance between their back walls corresponding to the width of the roasting kiln. An arch springs from the back walls of a pair of furnaces and on that is erected a block of shelf burners like that shown in Figs. 31 and 32 without the cellars for roasted ore, which in this case is dropped through chutes in the supporting arch to the floor between the distillation furnaces. The roasting furnace has a central chimney and the distillation furnaces each have two independent chimneys, the system of flues being so arranged that by an adjustment of dampers the products of combustion from the distillation furnaces can be discharged through the roasting furnace or directly into their own chimneys, which to a certain extent permits of a control over the firing.

The system of heating a roasting furnace by the flames discharged from a distillation furnace is to be condemned on several grounds. In the first place, both roasting and distillation are delicate operations in which a care-

ful adjustment of temperature is required and in distillation especially, which is an intermittent operation, the heat requirements vary considerably at different stages. Inasmuch as the processes of roasting and distillation do not march together in that respect, it is next to impossible to gauge the heat in the distillation furnace to that needed in the roasting furnace, or vice versa, and one of the processes must suffer, or what is more likely both will suffer. In the second place, the questions of air supply, gas velocity, avoidance of flue dust and loss of zinc by volatilization are even less under control than in a reverberatory furnace which is fired directly. As cleverly as the roasting furnaces are operated with the distillation furnaces at some places, with the advantage of long experience and intelligent supervision, the results as to degree of desulphurization in the roasting furnaces and percentage of recovery of metal in distillation are inferior to those obtained elsewhere, with the same ores and same management from independent furnaces, and more than offset the economy in fuel, which is not equivalent to the consumption of a separate roasting furnace, because more coal is burned on the distillation furnace grates than if retorts only were to be heated. There is an enormous waste of heat from any simple distillation furnace, it is to be conceded, but such waste heat is utilized better by restoring it to the distillation furnace itself, in the form of hot air, than by employing it for another purpose, which in all probability will interfere with the process of distillation and jeopardize the recovery of zinc that is the prime object of that process.

Capacity of Roasting Furnaces.—The capacity of roasting furnaces of all types is governed by the same principles. These are the quantity of ore exposed to free oxidation and the rapidity with which oxidation can be effected. Different types of furnaces vary in their manner of meeting these principles, but to a much less extent than is commonly supposed. In other words, the simplest hand raked reverberatory furnace, given the same ore, may do as efficient work in desulphurization as the most improved mechanical furnace; the advantage of the latter is chiefly in its economy of labor. Vice versa, a well designed mechanical furnace of any type should do as good work as a hand raked furnace.

The rapidity with which blende can be burned depends greatly upon its fineness of subdivision. The finer it is the more rapidly will it burn, assuming that air be given proper access to the particles, just as fine coal on a grate will burn more quickly than the large lumps, which continue to smoulder for a long time. Technical reasons, however, prohibit crushing the blende finer than 1 mm. and the material to be roasted is usually reduced only to about 2 mm. in size. The rate of roasting will depend then upon

the aggregate surface of the mineral particles exposed to free oxidation and the presentation of fresh particles as soon as those on top have been burned out. The latter consideration is the care of the roaster-man in a hand raked furnace; and of the metallurgist in one which is raked mechanically, it being assumed that the operation of the rakes once having been properly adjusted for a particular ore will ever afterward be correct. The great factor in determining the capacity of a furnace is therefore the hearth area. All other conditions being the same a furnace of 2000 sq. ft. of hearth should roast twice as much ore per 24 hours as one of only 1000 sq. ft. In assuming that conditions are the same in each case, it is implied that the ore remains an equal length of time in the furnace and is turned over with equal frequency, upon which of course the degree of desulphurization also depends. It is obvious that a furnace with six hearths, each 8×8 ft., from which a charge occupying one hearth is drawn every eight hours will roast no more ore than a furnace of only four hearths of the same dimensions, but in the former case the ore will have been in the furnace 48 hours and in the latter case only 32 hours. In each case three charges of the same weight would be withdrawn per 24 hours. The ore that had been 48 hours in the furnace would presumably be more completely desulphurized, although the gain might not be sufficient to compensate for the additional time. If, however, the furnace of six hearths be drawn every six hours, there will be obtained four charges per 24 hours instead of three, and the ore will have been 36 hours in the furnace, but in getting this greater output more labor will be required, because if one man has all that he can do to attend properly to a certain number of hearths, or square feet of hearth area, he cannot be expected to obtain the same efficiency from an additional surface. Hearth area, labor and time are therefore correlative in determining the capacity of a furnace. The time is dependent upon the character of the ore, its fineness of subdivision, the thickness of the bed lying on the hearth, the degree of desulphurization that is desired, the regulation of the air supply and the maintenance of the requisite temperature.

In roasting blende of 1 to 2 mm. size, it is usually spread over the hearth of the furnace in a bed from 2 to 4 in. thick. In mechanical furnaces the ore is fed and moved ahead continuously. In hand raked furnaces it is dropped on the first hearth in large charges at intervals of several hours. When a charge on the lowest hearth is drawn the ore on the hearths above is moved forward one step, thereby leaving room for a fresh charge on the first hearth. By "hearth" is meant here the working unit of the furnace. Thus a long reverberatory may be divided by imaginary lines or by actual steps into four hearths; similarly each floor of a double reverberatory may

be divided into two hearths. The ore is permitted to remain in the furnace from 15 to 36 hours. It is a common practice to draw a charge every eight hours; if the furnace has four hearths the ore in that case remains in it 32 hours. Six hearth furnaces are generally discharged every six hours, the ore having been roasting for 36 hours. Sometimes furnaces with two floors are drawn every 12 hours, the ore roasting for 24 hours. In Upper Silesia the ore is roasted for only 15 hours; a two-hearth reverberatory contains three charges at the same time; at the end of five hours a charge is moved forward to the second division of the upper hearth; five hours later it is dropped to the lower hearth and spread out over the whole of the latter, the bed of ore being consequently only half as thick as on the upper hearth. It is the most common practice, both in Europe and America, to charge every six or eight hours, leaving the ore in the furnace 32 or 36 hours. During the intervals the ore is raked every 15 to 30 minutes. The frequency of raking will depend of course a great deal upon the number of square feet of hearth that the operator has to attend to.

In the following chapter some results obtained practically in the roasting of blende in hand raked reverberatory furnaces are recorded. The data are unfortunately insufficient to enable a conclusive analysis, but it may be safely assumed that the ores roasted contain generally from 25 to 30% S, that they are in most cases crushed to between 1·5 and 2 mm. in size, and that they are burned down to approximately 1% S. Disregarding the difference in the rate of burning of various blendes and considering the chances of inaccuracy in some of the data gathered from numerous publications, it appears that the product of the number of square feet of hearth attended by one man and the pounds of raw ore roasted per square foot per 24 hours does not vary much, as appears in the following table:

Place.	Sq. ft. per Man.		Pounds per sq.ft.		Product in pounds.
Ammeberg...	224	×	25	=	5 600
Weir.....	333⅓	×	15	=	5 000
Lipine	103	×	54	=	5 562
Pittsburg....	512	×	11	=	5,632

It is logical to assume that the more area of hearth a man is required to attend to, the less frequently will the ore on any square foot of it be turned over; in other words, the lower will be the efficiency per square foot of hearth. In no other way can the great discrepancies in the quantity of ore roasted per square foot of hearth, as reported from various places, be satis-

factorily explained. It would appear to be a safe basis for calculation, in connection with hand raked reverberatory furnaces, fired with coal, that with 500 sq. ft. of hearth per man the quantity of ore roasted per square foot per 24 hours will be about 10 lb.; with 333·3 sq. ft. per man, about 15 lb.; and with 250 sq. ft. per man, about 20 lb. This is equivalent to saying that one man per shift of 12 hours can roast about 2500 lb. of ore, irrespective of the hearth area, within the limits mentioned. It appears, however, that with the use of natural gas as fuel one man attending to 500 sq. ft. of hearth can roast at the rate of 20 lb. per sq. ft., which also seems to be approximately the rate of the mechanical furnaces which are employed in the United States.

Loss of Weight in Roasting.—A sulphide ore suffers a loss of weight in roasting, which is generally proportional to its tenor in sulphur, because oxygen (which is an element of a lower atomic weight) is substituted for the sulphur. In so far as the metallic sulphides are converted into sulphates, however, there is an increase in weight. The actual loss in weight which an ore will experience can be determined only by experiment, but the approximate result can be calculated from the analysis of the raw ore.

For example, assume an ore assaying 60% Zn, 29·36% S and 10·64% SiO_2. One metric ton of the ore would contain 600 kg. of zinc, 293·6 kg. of sulphur and 106·4 kg. of silica, or expressed mineralogically 893·6 kg. of blende and 106·4 kg. of quartz. If this ore were burned completely, the sulphur being entirely eliminated and its place taken by oxygen, the 600 kg. of zinc would be combined with 146·8 kg. of oxygen instead of 293·6 kg. of sulphur. The weight of the zinc oxide would be 746·8 kg. The quartz would undergo no change, so the weight of the roasted ore would be 746·8 kg.+106·4 kg.=853·2 kg. The loss of weight would be 1000 kg.−853·2 kg.=146·8 kg.=14·68%. The percentage of zinc in the roasted ore would be 600÷853·2=70·32.

In the case of so simple an ore as that estimated above, the loss in weight by a complete roasting will always be one half the tenor of sulphur. Thus if the ore assay 50% Zn and 24·46% S, the remainder being quartz, the loss in weight will be 0·5×24·46=12·23%; and an ore assaying 40% Zn and 19·57% S will lose 0·5×19·57=9·785% in weight.[1]

Practically the sulphur is never completely eliminated from an ore. Assuming that in the case of the ore assaying 60% Zn, 29·36% S and 10·64% SiO_2, a half unit of the sulphur fails to be burned off, the composition of the ore after roasting will be as follows:

[1] These percentages are calculated on the assumption that the atomic weight of oxygen is 16, sulphur 32, and zinc 65·4.

5·00 kg. S combined with	10·26 kg. Zn	= 15·26 kg. ZnS
589·74 kg. Zn "	144·26 kg. O	= 734·00 kg. ZnO
Unaltered silica		= 106·40 kg. SiO_2
Weight of roasted ore		855·66 kg.
Loss of weight		= 144·34 kg.
Weight of raw ore		= 1000·00 kg.
Percentage of zinc in roasted ore	600 ÷ 855·66	= 70·12
Percentage of sulphur in roasted ore	5 ÷ 855·66	= 0·58

The retention of so small a quantity of sulphur as a half unit does not materially affect the loss in weight.

Generally zinc sulphide ores contain, besides blende and quartz, small quantities of ferrous monosulphide isomorphous with the blende; also, pyrite, galena, calcite and dolomite, all of which are changed by roasting. If sulphur and carbonic anhydride be completely eliminated:

1 kg. FeS	becomes	0·909 kg. Fe_2O_3
1 kg. FeS_2	"	0·667 kg. "
1 kg. PbS	"	1·268 kg. $PbSO_4$
1 kg. $CaCO_3$	"	0·560 kg. CaO
1 kg. $MgCO_3$	"	0·476 kg. MgO

Upon the sulphur combined with zinc there is a loss of weight of one half; upon that combined with iron as FeS, one quarter; upon that combined with iron as FeS_2, five eighths; the sulphur combined with lead on the other hand leads to a threefold increase in weight. The loss in weight of a mixed sulphide ore can be easily approximated from these data.

Given an ore assaying 45% Zn, 10% Fe, 6% Pb, 32·10% S and 6·9% SiO_2. The lead requires 0·93% S to exist as PbS; and the zinc 22·02% for ZnS. This leaves 9·15% S, which must be combined with the iron. If the iron were all present as monosulphide only 5·71% would be required, wherefore it is evident that 9·15 − 5·72 = 3·43% S must be with iron as an extra atom, i.e., there must be 6·86% S in the form of FeS_2 and 9·15 − 6·86 = 2·29% in the form of FeS. The composition of the ore before and after roasting, assuming the latter to be complete, will be as follows:

Composition of Raw Ore.			Factor *a*	Roasted Ore
Mineralogical	Chemical			
67·02% ZnS	= 45% Zn + 22·02% S		0·500	45·0 + 11·01*b*
6·93% PbS	= 6% Pb + ·93 "		3·000	6·0 + 2·79*c*
6·29% FeS	= 4% Fe + 2·29 "		0·750	4·0 + 1·72*b*
12·86% FeS_2	= 6% Fe + 6·86 "		0·375	6·0 + 2·57*b*
6·90% SiO_2	= 6·9% SiO_2	....		6·9 ....
100·00	67·90	32·10		67·90 18·09
	100·00			85·99

a Factor with which to multiply weight of sulphur to obtain weight of replacing oxygen or SO_4. *b* Oxygen. *c* SO_4.

The loss of weight is therefore 100−85·99=14·01%. The percentage composition of the roasted ore is calculated by dividing the weight in kilograms of its constituents by 859·9.

Loss of Zinc in Roasting.—Loss of zinc may occur in roasting in two ways: (1) By volatilization; (2) by the production of dust which is carried away by the draught. The metal which is lost in the former way cannot be recovered, at least not economically; that which is lost in the latter way can be largely regained by causing the dust-laden gas to pass through a dust-settling chamber of suitable design and dimensions.

Fume.—The loss of zinc as fume in roasting blende in reverberatory furnaces is due partly to the direct volatilization of zinc sulphide and oxide, which, according to experiments of Percy[1] and Stahlschmidt,[2] are perceptibly volatile at high temperatures, and partly to the reduction of zinc oxide, formed from the blende, by solid carbon (soot) and carbon monoxide in the gases of combustion passing over the ore. There is evidence that the reduction of zinc oxide may begin at a temperature below the boiling point of zinc, which is a good deal below the maximum frequently attained in the furnace roasting blende. The zinc vapor thus produced is carried off and is immediately reoxidized by the carbon dioxide and excess of air present in the gases of combustion. This fume cannot be settled in any ordinary system of dust chambers and is lost through the chimney. The roofs of roasting furnace buildings often show a white coating of zinc oxide, which has been deposited from the fume dispersed into the atmosphere.

If the temperature in the roasting furnace does not much exceed 900° C. the loss of zinc by the direct volatilization of sulphide and oxide is comparatively slight. The loss by reduction and volatilization is more to be feared, since that may begin at 900° C., or a few degrees higher.[3] It follows therefore that the loss of zinc is likely to be higher in roasting in reverberatory furnaces, in which the ore is directly exposed to the gases from the fireplace, than in muffle furnaces wherein no carbonaceous material comes in contact with the oxidized ore. In either type of furnace the loss is minimized by a careful regulation of the temperature.

Dust.—The loss of zinc as dust is purely mechanical, arising from the fine particles of ore which are carried off by the draught especially during crushing of the ore and when it is fed into the furnace. This loss is also greater in reverberatory furnaces than in muffle furnaces, since there is a much greater volume of gas to be carried over the hearth of the former, and

[1] Metallurgy, edition of 1861, p. 540.

[2] Berg- u. Hüttenm. Ztg., 1875, p. 69.

[3] Refer to p. 198. The loss by reduction and volatilization which is actually experienced in blende roasting was investigated by Hasenclever (Berg- u. Hüttenm. Ztg., 1875, p. 69).

although the cross section of the roasting chamber is always made larger than in muffle furnaces the velocity of the gas is nevertheless likely to be greater. In reverberatory furnaces, moreover, the ore-feed is showered on the hearth directly in the stream of escaping gas, while in muffle furnaces the gas is usually taken off a little in advance of the ore-feeding opening. It is obvious that in either type of furnace the production of dust will be greater the finer is the material to be roasted.

The dust carried over from a roasting furnace can be recovered to a large extent by settling in outside dust chambers, the efficiency of which depends upon the degree to which the velocity of the gas, by which the dust is entrained, is checked, together with the extent and nature of the surfaces to which it is exposed. The requisite retardation of the current is accomplished by turning it through a chamber of such increased sectional area that the volume of gas exhausted from the furnace will travel through it at a much diminished velocity (the action of the dust chamber corresponding to that of a lake into which a river flows at one end and out at the other, much of the silt carried into the lake being deposited in its quiet water); or by diminishing the velocity of the current by friction, either by increasing the length of the flue or by exposing a greatly increased surface to the dust-laden gas, as is done by the system of Freudenberg plates. The system of filtration through woolen bags, which is applied to certain kinds of smelting furnaces, is inapplicable to roasting furnaces, even after cooling the gas, because of the sulphuric anhydride which is always present in it and would soon destroy the bags.

Design and Proportions of Dust Chambers.—Dust chambers are frequently to be seen in which the gas is made to pursue a zigzag course, over one partition and under the next one. This is wrong in theory and inefficient in practice. Instead of a placid lake of almost stagnant gas, in which the dust can settle quietly into a compartment where there is no draught at all, there is in a zigzag flue a fierce draught that sweeps the dust along with it and the only chance that the latter has to settle is what is afforded by the retardation in the velocity of the current because of the increased friction due to its labyrinthine channel and the acceleration by gravity of the heavier ore particles when projected downward, which having been thus deposited may escape being taken up again by the draught.

Leaving out of consideration the costly systems of Freudenberg plates, Roesing wires, etc., the simplest and most effective dust chamber is an enlarged flue, with transverse walls rising one or two feet above the floor to prevent the deposited dust from drifting along the latter. Between the upper edges of those walls and the arch of the flue there must be a sufficient

vertical sectional area to permit the gas to move slowly. The dust particles drop out of the gas according to their weight into the compartments in the bottom of the flue, where there is no current at all and they remain undisturbed until removed through doors in the sides. If desired the floor of the dust chamber may be made Λ shape, so that the dust will discharge directly into a car alongside, but since that increases the cost of the dust chamber, reduces the capacity of the dead compartments and even in large works the quantity of dust to be handled in tons is not very great, the flat floor built directly on the ground is on the whole the preferable arrangement.

The dimensions which a dust flue should have are readily determined approximately from the volume of gas that must pass through it, the method of computing which has been described in a previous section of this chapter. It was estimated there that the sulphurous gas from a muffle furnace might under certain conditions of practice amount to 122,000 cu. ft. at 0° C. per ton of ore assaying 60% Zn and 30% S. The volume of gases is proportional to their absolute temperature, which is expressed by the formula

$$V_1 = V_0 \frac{T}{t}.$$

The absolute temperature on the centigrade scale is —273°. If the gas were discharged from the furnace at 500° C., T would be 773° C. and t being 273° C. the volume of the escaping gas would be $(773 \div 273) \times 122{,}000 = 345{,}260$ cu. ft. A furnace roasting 10 tons of ore per 24 hours would therefore discharge 3,452,600 cu. ft. of gas at 500° C. The discharge per minute would be $3{,}452{,}600 \div 1440 =$ approx. 2400 cu. ft. $=40$ cu. ft. per second. Assuming that the dust will settle quickly from a current of 4 ft. per second, which is only one third the velocity corresponding to a light draught on Beaufort's scale,[1] the dust chamber must have a free area in vertical cross-section of $40 \div 4 = 10$ sq. ft.

In roasting 10 tons of ore per day in a reverberatory furnace the volume of combustion products at 0° would be $10 \times 340{,}000 = 3{,}400{,}000$ cu. ft., which at 500° C. would be $(773 \div 273) \times 3{,}400{,}000 = 9{,}622{,}000$ cu. ft. $=$ approx. 112 cu. ft. per second. In order to permit that volume of gas to pass at a velocity of 4 ft. per second it would have to be given a free cross-sectional area of $112 \div 4 = 28$ sq. ft. If the temperature of the gas were

[1] Four feet per second=a little less than three miles per hour. Dust will settle quickly from a current of that low velocity. Dr M. W. Iles, who has devoted much study to this problem, recommends in his recently published treatise on Lead Smelting, p. 155, that dust chambers should be designed for a velocity of 200 to 400 ft. per minute (3·33 to 6·67 ft. per second), and the upper limit should never be exceeded. Having especial reference to lead smelting practice he remarks that dust flues are seldom constructed large enough. In acid works the flues are often designed for very low velocities.

273° C. the volume discharged per second would be approximately 80 cu. ft., requiring a flue area of 20 sq. ft.[1]

These estimates do not take into account the effect of friction between the gas and the walls of the flue and being based on the average volume of gas per 24 hours do not allow for variations in the quantity, which it is important to do.

Space will not permit a discussion in this treatise of the numerous methods of dust and fume collection other than by simple chambers, including the Roesing wires, Freudenberg plates and the excellent system of Cowper flues, which are of importance to the lead smelter, who is confronted by the difficult problem of condensing fume as well as depositing dust. In roasting blende on the contrary the production of fume can be kept down to an insignificant figure and the problem of recovering lost value chiefly concerns the dust only. It alone is worthy, however, of the most attentive consideration. If in roasting 10,000 tons of ore per annum only 1% were lost as dust, which could be recovered, there would be a saving of 100 tons of ore, worth perhaps $2500; that sum would be the interest of $25,000 at 10% or of $12,500 at 20% and consequently well worth the expenditure for the addition to plant that would be required to get it.

The length of a dust chamber is an important factor, since the dust must not merely be given an opportunity to settle by reducing the velocity of the current, but also it must be given time to do so. The length of a dust chamber through which the movement is produced by the suction of a chimney is limited, however, to that whereof the radiation will not reduce the temperature of the gas below what is required to effect the necessary draught of the chimney. This condition does not impose a quick limitation, because as will be shown in the section on chimneys in the chapter on "Fuel and Systems of Combustion," differences in temperature between 200° and 400° C. do not greatly affect the draught. There is not therefore the same necessity of preventing radiation from the dust chamber of a reverberatory furnace that there is from that of a muffle furnace. As the gas cools its volume decreases, wherefore its velocity will vary correspondingly; it is sufficient to base the dimensions of a flue for a reverberatory furnace on the mean temperature between the furnace and the chimney.

The dust-settling system of a muffle furnace is limited by the temperature

[1] These temperatures are often greatly exceeded. Iles noted a temperature of 262° C. in a flue leading from two mechanical roasting furnaces, 100 ft. from the latter. During a number of years the temperature at the base of two brick chimneys, drawing from roasting furnaces, ranged from [illegible]° to 399° C. In the main flue from mechanical furnaces 760° C. was observed; at the entrance into the dust chamber of a hand-raked furnace, 900° C.! (Lead Smelting, pp. 155 and 173.)

which must be preserved for the economical operation of the Glover tower, wherefore loss of heat by radiation must be avoided so far as possible by thick brick walls. In any case, the work of the short dust chambers ordinarily connected with blende roasting furnaces consists only in the collection of the dust mechanically carried over. Such zinc as has been volatilized and oxidized is in too fine a state of subdivision to be collected in any way short of bag filtration, and goes up the chimney of the reverberatory furnace or to the Glover tower of the muffle furnace, appearing finally in the sulphuric acid; fortunately the zinc lost by volatilization from a muffle furnace is insignificant.

Technical Results.—The percentage of dust carried away from hand raked reverberatory furnaces is likely to be 1 to 3% of the weight of the raw ore charged in. It appears to be no higher with the long mechanical furnaces, the quiet, regular mechanical movement of their rakes seeming to raise no more dust than the more violent, if less frequent, raking by hand. In roasting copper ores, which are generally crushed less finely than blende and remain a shorter time in the furnace, such low percentages of dust as 0·5 and 0·8 have been reported. With revolving cylinders, like the Brückner, the production of dust is much greater, ranging from 3 to 5%. With proper settling chambers practically all of this dust can be collected.

In roasting with shelf burners at several of the smelteries at Pittsburg, Kansas, the total loss of zinc was considered to be from 1 to 3%, averaging 2·25 to 2·50%; at Collinsville, Ill., according to Horace F. Brown, in a private communication, it used to be about 2%, with the same type of burners; in both cases the ores averaged 56 to 60% Zn before roasting, and there were no dust chambers. These figures agree with what is stated by Kerl and other writers as to similar practice in Europe. Sometimes the loss is kept down to a very low figure. At Oberhausen, Westphalia, according to Mahler,[1] it did not exceed 0·75%, roasting in a two-hearth reverberatory. This loss was due almost entirely to dust, not to volatilization. The ore assayed 42 to 56·4% Zn, with 25 to 26% S, and was burned down to between 0·57 and 0·83% S. The employment of the sulphurous gas from muffle furnaces for the manufacture of sulphuric acid compels the installation of means for the separation and recovery of the dust, because aside from the value of the dust it is highly objectionable in the sulphuric acid process.

[1] Ann. des Mines, 1885, VII, p. 152.

IV.

ROASTING FURNACES.

In roasting zinc blende the same kinds of furnaces are available as for other sulphide fines, and probably a greater variety is in use for that purpose at the present time than in any other branch of metallurgy, while there are numerous furnaces that are not now employed for blende roasting which are possibly adaptable to that purpose. For this reason some of the more modern which have not yet been used by zinc smelters will be described herein, together with those which have been. The diversity in the mechanical furnaces in use in the United States is due to the fact that most of them are patented and the patentees have sold territorial rights to certain zinc smelters, which has prevented any one furnace from coming generally into use, and for the more part has limited the use of each type of furnace to one smelter. Disregarding heaps and stalls, which are seldom employed nowadays, the roasting furnaces in use for desulphurizing blende, together with a few which are perhaps adaptable to the purpose but have not yet been applied to it, are most conveniently classified as follows:

I Shaft furnaces.
  a Kilns and burners, suitable only for lump ore.
  b Gerstenhöfer kilns, suitable for fine ore.

II Reverberatory furnaces.
  a Raked by hand.
    1 Furnaces with one long hearth (Fortschaufelungsöfen).
    2 Furnaces with two superimposed hearths (Freibergeröfen)
    3 Furnaces with more than two superimposed hearths.
  b Raked mechanically.
    1 Furnaces with stationary hearths.
    2 Furnaces with movable hearths, other than revolving cylinders.
    3 Revolving cylinders.
  c Automatic furnaces.

III Muffle furnaces. (including furnaces used for burning pyrites for sulphuric acid manufacture, which are commonly run without extraneous firing, but can be readily converted into muffle furnaces.)
  a Raked by hand.
  b Raked mechanically.
  c Revolving cylinders.

There are some furnaces in which the principles of two or more types are combined, for example, hand-raked furnaces with two long hearths of which

the upper is a muffle and the lower a reverberatory. The Hasenclever & Helbig furnace was a combined muffle and reverberatory, while the movement of the ore through it was effected partly by hand raking and partly automatically by gravity.

ROASTING IN HEAPS AND STALLS.—Zinc blende was partially desulphurized at certain places in the past by roasting in heaps and stalls, but at present there is no place where that is done except at Langelsheim and Astfeld in the Lower Harz, Germany, where the mixed sulphide ore (blende, galena and pyrite) of the Rammelsberg is roasted in heaps to convert the zinc sulphide into sulphate, which is then removed by lixiviation. This process is described in Chapter XIX.

In roasting a mixture of blende and pyrite in heaps or stalls the oxidation of the former is assisted by the heat generated by the freely burning pyrite, but even then the process is very incomplete in so far as the blende is concerned, because the fragments of that mineral soon become coated with a crust of oxide and sulphate, of which a very thin shell checks further combustion and the product of a stall in which the pyrites has been well burned will show the fragments of blende with large kernels of unaltered sulphide. By breaking the blende to a comparatively small size before charging into the stall the oxidation can be naturally carried further, and I have in that manner succeeded in burning blende, mixed with pyrite, to a sufficient extent to permit the mixture to be run successfully in a matte smelting blast furnace, the slag made being high in ZnO, while the quantity of raw blende left in the ore when roasted in large pieces was too great to make smelting feasible.

Zinc blende alone cannot be successfully burned in stalls. After the charge is fired with a small quantity of split wood, in the usual manner of starting a stall roast, the blende will burn actively for a day or two and will then gradually die out and after smoldering feebly for several days combustion will cease entirely. Upon discharging the stall the lumps of blende will be found coated with white and gray zinc oxide and sulphate, a good deal of which will crumble off in handling the material and can be removed by screening. The lump blende if recharged into the stall can be ignited again and a second crust of zinc oxide and sulphate may be formed, and by repetition of the process a good deal of the blende may be oxidized. I have referred to the subject at this length because the roasting of the Rammelsberg ore in the Lower Harz is conducted in that manner, with the difference that it is done in heaps and the ore contains pyrite.

Blende was formerly roasted in Pennsylvania in heaps of 8·5 m. length, 4·6 m. width, and 2·5 m. height, the ore being supported on iron bars, below

which a wood fire was maintained; the desulphurization was completed by crushing the heap roasted ore and calcining in a reverberatory furnace.[1] Stalls fitted with movable grate bars were formerly used at Letmathe, in Westphalia, for burning blende from Iserlohn, the roasting being finished in a reverberatory furnace. The stalls were roofed and the roast gas was used for sulphuric acid manufacture.[2] They partook rather of the nature of kilns (q. v.) than of stalls proper.

Shaft Furnaces.

Shaft furnaces, which as a class comprise kilns, pyrites burners and Gerstenhöfer furnaces, are used at several works in Europe for the desulphurization of blende without fuel, but their action is incomplete and the roasting has to be finished in reverberatory furnaces; because of that drawback the employment of this type of furnace is limited. Kilns can be used only for lump ore; pyrites burners will burn a finer size; while the Gerstenhöfer and similar furnaces are adapted only to fines.

Kilns.—These are high shaft furnaces, either with a grate or a saddle form of hearth. It is doubtful if any of this type of furnaces is to be seen in use for blende roasting at the present time outside of Freiberg, Saxony, and certain works in Upper Silesia. They are generally rectangular brick shafts 1·5 to 4·5 m. in height, 1 to 1·5 m. in width and 1·5 to 2·5 m. in length, inside measurements. The ore having been ignited burns of itself, but the combustion will rarely continue after the percentage of sulphur has been reduced to eight.

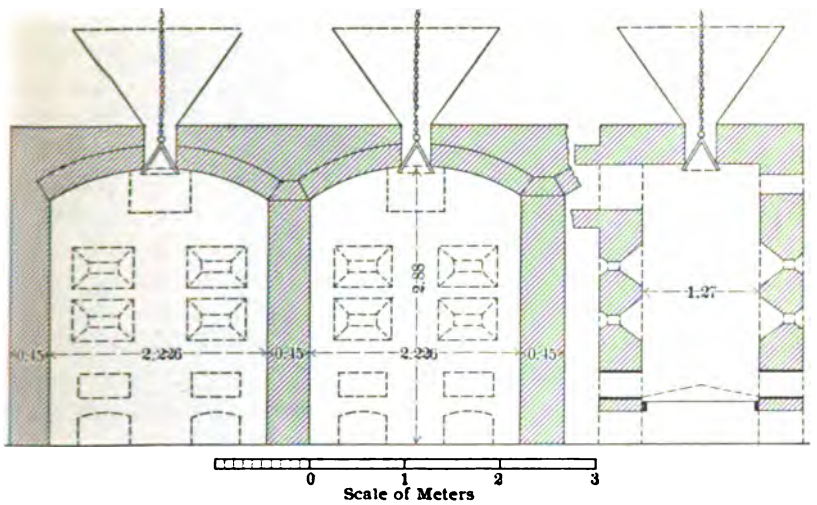
Kilns used for blende roasting at Freiberg, Saxony, are illustrated in the engravings on the next page, which require no special explanation beyond remarking that the working doors in the front and back serve for the admission of bars to keep the charge of ore open and regulate its even descent; the flue which carries off the sulphurous anhydride gas leads into a larger flue, common to a bench of kilns. In 24 hours each kiln of the dimensions shown in the engravings will burn down to 8% S about 1·2 metric tons of ore assaying originally 30% S. The roasted ore is drawn four times during the 24 hours. It is crushed and roasted further in reverberatory furnaces.

Burners.—These are low, grate-kilns, such as are frequently employed for burning pyrites for sulphuric acid manufacture. They will burn blende of bean to walnut size down to about 6 to 8% S. Probably they are to be

[1] Berg- u. Hüttenm. Ztg., 1872, p. 53, p. 61.
[2] Kerl, Grundriss der Metallhüttenkunde, p. 439.

found nowhere in use at the present time outside of those works in Upper Silesia which make sulphuric acid.

At Lipine, Upper Silesia, the burners are square shafts of 1·25 m. sides and 2·8 m. height, the thickness of the column of blende over the grate being only 0·4 m. One of these burners will in 24 hours put through 500 kg. of blende assaying 25% S and yield a product with 10% S, the roast gases containing 6% SO_2 by volume. One man attends to 10 burners per 12-hour shift, and there are 26 burners in a block. At the Reckehütte, near Rosdzin, in Upper Silesia, the burners are 1 m. square and 2·5 m. in height,



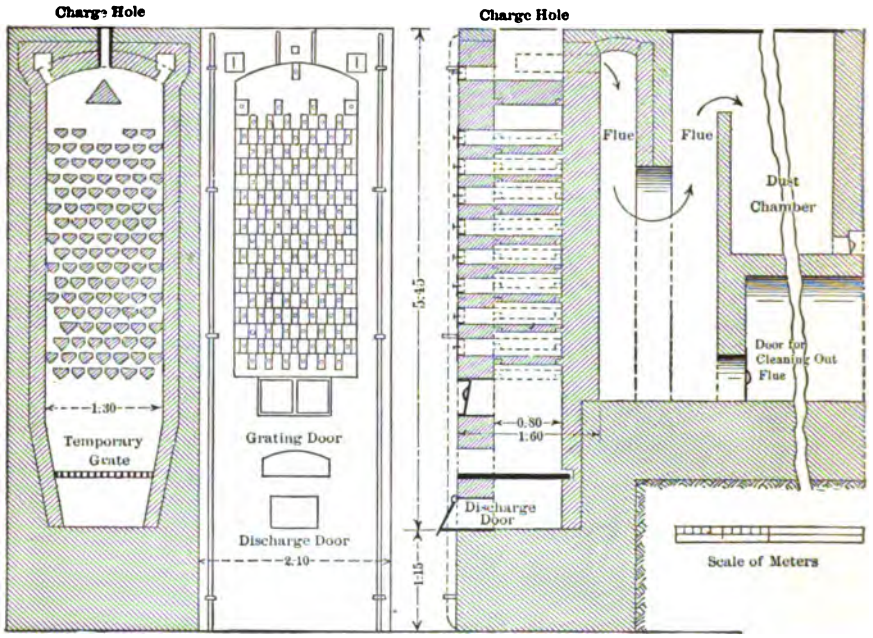
FIGS. 21 AND 22. KILN USED AT FREIBERG, SAXONY.

the column of blende over the grate being 0·6 m. The blende is charged nut size and with a sulphur tenor of 24 to 33% In 24 hours a burner desulphurizes 350 kg. down to 7% S, while the gas contains 7% SO_2 by volume. From 23 to 30 burners are united in a block, and a block is managed by two roasters and two helpers per day, a total of four men.[1] As at Lipine, the kiln-burned ore is further desulphurized in reverberatory furnaces.

GERSTENHÖFER KILN.—The Gerstenhöfer kiln is the only successful design of shaft furnace for roasting fine ore (blende) without fuel. It is essentially a shaft 5·5 to 6 m. high and about 1·4×0·8 m. in horizontal section, which is filled with numerous horizontal bars of fire brick, set stag-

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 43.

gered, as shown in the accompanying engravings. Commonly there are 15 or 17 tiers of bars, with six and seven alternately in a tier. The ore, fed into the shaft through holes in the roof, spreads out over these bars, sliding off at an angle of about 33°, and burns with the air admitted through a flue leading into the bottom of the shaft. As more ore is fed in at the top, it is gradually pushed off from the lower shelves and is finally delivered at the bottom of the shaft as completely burned as can be effected



FIGS. 23 AND 24. GERSTENHÖFER KILN.

Fig. 23: Transverse vertical section and front elevation. Fig. 24: Longitudinal vertical section through center.

without the employment of other fuel. The movement of the ore goes on continuously according to the rate of the feed.

There are various forms of charging apparatus for regulating the feed of the ore and avoiding the production of an excessive quantity of dust when the ore is dropped into the furnace. Openings are arranged in the front wall to remove obstructions to the passage of the ore and insert new bars when necessary. The lower part of the shaft must also be arranged with a grate to heat the kiln when first put in operation. The roasted ore may be shoveled from the bottom, or it may be made to fall into a hopper under

which a car can be put, or it may be discharged continuously by some simple mechanical device. At the best, the percentage of dust escaping from the Gerstenhöfer kiln is high and the sulphurous gases must be conducted through a dust chamber wherein their velocity will be checked sufficiently to allow as much as possible to drop out.

The Gerstenhöfer furnace is in use for blende roasting only at Freiberg, Saxony, and at Swansea, Wales. In the most favorable cases it will put through 1 to 1·2 metric tons of ore per 24 hours, which may be burned as

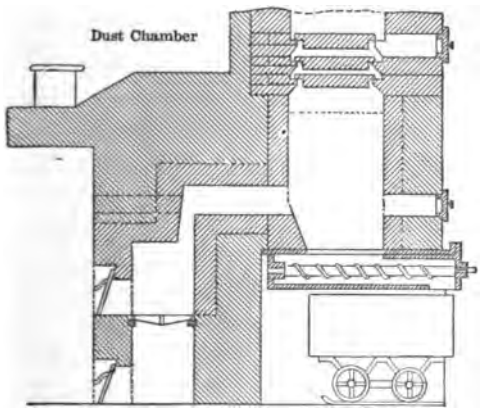


FIG. 25. GERSTENHÖFER KILN WITH FIREPLACE AND MECHANICAL DISCHARGE.

low as 5% S, no carbonaceous fuel being used. The furnace can be arranged, however, with a fireplace on one side of the shaft, from which the flames draw into the shaft and up between the bars of the latter. With such an arrangement the ore may be burned down to 1% S, but the roast gases are of course impaired for sulphuric acid manufacture.

In putting the Gerstenhöfer furnace in operation, 15 or 16 grate bars are put in and firing is begun gently, so as to heat gradually and prevent the shelves from cracking. The gases of combustion are not allowed to enter the flue leading to the lead chambers. After the furnace has been raised to bright red heat, feeding is begun, firing being continued until it is observed that about the fourth row of shelves from the bottom is beginning to receive ore; the grate bars are then drawn out. The admission of air into the furnace must be carefully regulated. If too much is admitted the fire creeps up in the shaft; if too little, it goes downward. In the former case the furnace is too cold, and in the latter too hot.

Reverberatory Furnaces.

The general principles governing reverberatory roasting furnaces were discussed in the previous chapter. They apply equally to all types of this class of furnace—hand raked, mechanically raked, revolving cylinders and automatic.

Reverberatory Furnaces, Raked by Hand.

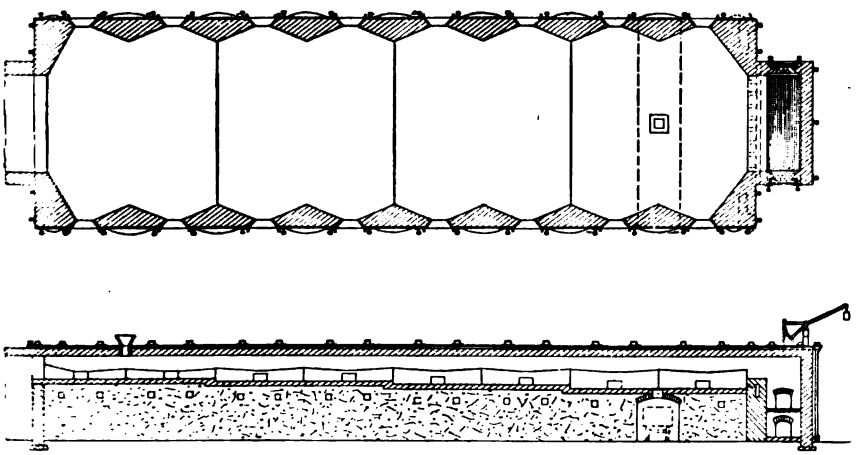
Reverberatory furnaces, raked by hand, are the oldest type of furnaces for the dead roasting of fine blende, and in proportion to their hearth area (capacity) they are the cheapest to construct. On the other hand, they are much less economical in labor than are mechanical furnaces, and as compared with muffle furnaces their sulphurous gases are not only useless but also are a nuisance; the laws of certain countries require that such gases be rendered innocuous, which it is almost always costly to do. The advantages of the reverberatory furnace, worked by hand, are outweighed by those of the mechanical and muffle furnaces, and consequently they are being gradually displaced, although they are still extensively in use.

Reverberatory furnaces, worked by hand, may be built with one long hearth (Fortschaufelungsöfen), with two hearths (Freibergeröfen, Silesian furnaces, etc.), or with three or more superimposed hearths. The design of the various types is governed by the same general principles. It is to be remarked that the reverberatory furnace with a single, long hearth is seldom used for blende roasting, the two-hearth furnace being most generally employed in Europe and the shelf burners in the United States.

General Principles.—The length of hearth of a reverberatory roasting furnace, with a single fireplace at one end, is determined primarily by the capacity of the ore to develop heat by its oxidation, i.e., the length of hearth that may be advantageously employed depends chiefly upon the tenor of sulphur in the ore. Peters states,[1] referring especially to pyritous ores, that the fire on the grate is usually incapable of maintaining the requisite temperature for more than 30 to 35 ft. beyond the bridge; and that an ore with 15% S will furnish only enough heat for a hearth 32 ft. long, while with 20% S the length may be increased to 48 ft. and with 25% S to 64 ft. Besides the generation of heat, however, an important consideration is its conservation, which appears to be better secured in the double hearth furnace than in the single, wherefore the total hearth length of the former may be somewhat greater than in the case of the latter. The experience in roasting blende containing 25 to 30% S seems to have prescribed 40 ft.

[1] Modern Copper Smelting, 7th ed., p. 176.

(12 m.) as the proper length for a single hearth furnace, while double hearth furnaces have been built with a total hearth length of 48 ft. (about 7·25 m.). These dimensions are considerably exceeded in certain muffle furnaces (e.g., the Rhenania), in which the heat of the burning sulphur is carefully conserved, while the air for its oxidation is more or less pre-heated and only a comparatively small excess is admitted into the roasting chamber. The width of a reverberatory roasting furnace is governed solely by the ability of the operators to work the charge in proper manner. Experience has shown that 16 ft. is the maximum for a furnace with doors on both sides, and 8 ft. when there are doors on only one side.



FIGS. 26 AND 27. HAND RAKED REVERBERATORY FURNACE, 14×60 FT.
Fig. 26: Plan of hearth. Fig. 27: Vertical, longitudinal section.

Reverberatory furnaces with wide hearths and working doors on each side, such as are used in connection with lead and copper ores, are very rarely, if at all, employed for blende roasting. The conventional furnace for the latter purpose, both single and double hearth, has a narrow hearth, with doors only on one side, but two furnaces are commonly united in a block, back to back, so that the result is practically a wider hearth divided by a longitudinal wall. This system, which also finds application in the Rhenania muffle furnace, is thought to diminish the cooling effect of the false air admitted while raking the ore and probably does so.

The grate area of a blende roasting furnace is determined by the hearth area and the character of the fuel to be burned, or rather upon the rate of combustion that must be effected. Experience indicates that in roasting

in reverberatory furnaces with good coal, like the Belgian and Westphalian, blende assaying approximately 25% S can be roasted with a coal consumption of not more than 25% of the weight of the raw ore. The roasting of 2000 lb. of ore would then require 500 lb. of coal, which if burned at the rate of 5 lb. per sq. ft. of grate per hour would require approximately 4 sq. ft. of grate. If the ore were roasted at the rate of 20 lb. per sq. ft. of hearth per 24 hours, 2000 lb. of ore would require 100 sq. ft. of hearth and the ratio of hearth to grate area would be 25 : 1. There must be sufficient chimney draught to burn 5 lb. of coal per sq. ft. of grate per hour. Practically there is a wide variation in the rates of combustion per square foot, both of ore and of coal, in the percentage of coal consumed and in the ratio between hearth area and grate area.

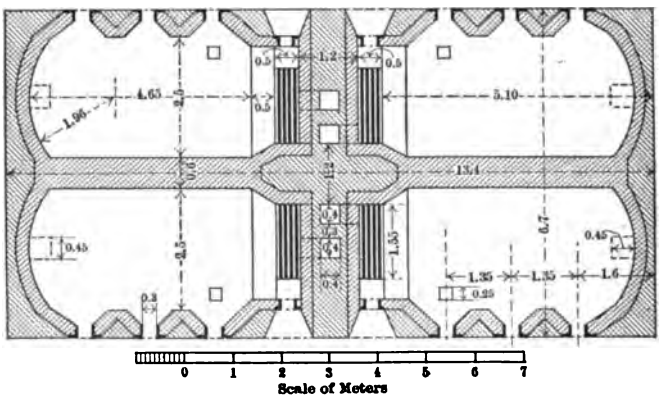


FIG. 28. BLOCK OF FOUR REVERBERATORY FURNACES.
Plan of Hearths.

The number of working doors per side should be as few as possible, but the maximum limit, center to center, is about 8 ft. Generally, the doors are set at shorter intervals than that, e.g., 7 ft., 6 ft., or 5 ft. It is a common practice to provide the door frames with lugs, so that they will be held firmly in place by the buckstaves, which will be so arranged that one will come on each side of a door opening. The bottom of the door frames should set flush with the hearth. The doors themselves should be of a kind that can be closed tightly. In connection with the double hearth reverberatory furnaces employed in Missouri and Kansas cast-iron doors, swinging from hinges on one side, are commonly adopted.

The walls of a reverberatory furnace should be at least 13·5 in., or one and a half brick, in thickness. The arch should be 9 in. thick, the brick

being laid as headers; radiation of heat from the arch may be reduced by extending the side walls to the height of the crown and filling in the spandrels with sand. The hearths may be laid with fire brick tiling, or with standard fire brick, a single course being set edgewise. The common practice of filling in portions of the hearth room between the working doors with projecting, triangular masses of brick work is not to be recommended, because it is useless; with properly designed working doors there is not much inaccessible space between them, and such as there is becomes permanently filled with ore, which does no harm by remaining there.

SINGLE HEARTH FURNACES.—The height of the roasting chamber, from hearth to roof, at the fire-bridge end should not exceed 30 in., and it is advisable to concentrate the heat further on in the furnace by raising the hearth toward the chimney end, either by steps or by a gradual inclination. The steps in the hearth may be conveniently made 3 in. high. This method

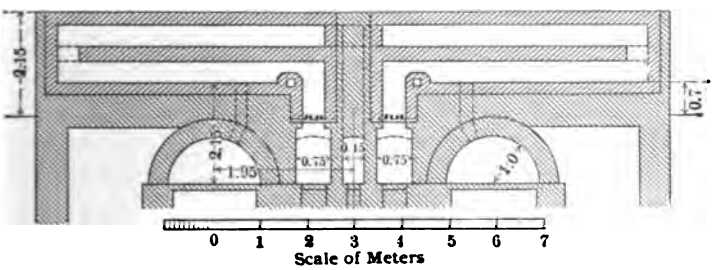


FIG. 29. BLOCK OF FOUR REVERBERATORY FURNACES.
Longitudinal section through center of hearth.

is preferable, on account of structural reasons, to a depression of the arched roof of the furnace. Sometimes the arch is depressed over the first section of the hearth, so as to throw the flames down on the ore; and frequently the hearth room is made low throughout, the height from the sole to the center of the arch being as little as 18 in. In view of the modern ideas as to the advantage of heating by radiation, however, it would appear to be the more correct design to provide room for a large volume of luminous flame at the fire-bridge end of the furnace, and bring the hot gases down to the ore further on, where the heating must be chiefly by contact.

The level of the hearth should be, for the convenience of the operators, precisely 3 ft. above the floor of the furnace house, and the latter should be sloped to correspond with the elevation of the hearth. The height of the fire bridge above the hearth is usually 9 to 12 in.; and above the grate, 15 to 20 in., but the latter dimension varies considerably according to the

kind of coal and the method of burning it (whether in a thick or a thin bed). The fire bridge is usually made 18 to 24 in. thick; it is a good design to arrange for cooling it by a horizontal air canal, which may be advantageously connected by ports with the roasting chamber, thus supplying the ore with fresh and partially preheated air. The grate itself may be an ordinary, horizontal grid, or a step grate, or the furnace may be fired with producer gas, natural gas, or crude petroleum. The design of the fire box is discussed at more length in Chapter VIII.

DOUBLE HEARTH FURNACES.—A good deal of what has been said with respect to the single hearth reverberatory applies also to the double hearth, which may be in fact considered a single hearth furnace cut in halves, with one half superimposed upon the other. This arrangement is considered to possess the advantages of better conserving the heat and economizing floor

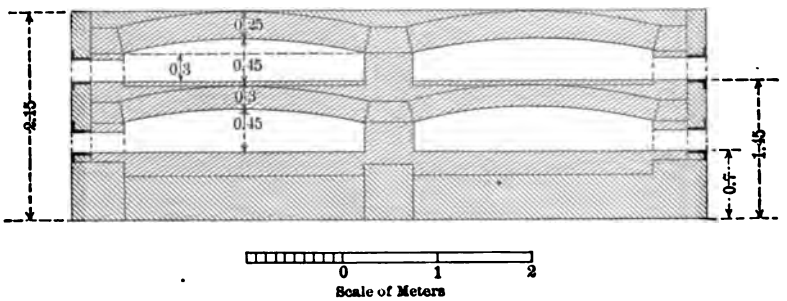


FIG. 30. DOUBLE HEARTH REVERBERATORY FURNACE.
Transverse vertical section.

space and roof. It has the disadvantages of less ease in making repairs and less convenience in working. Either a movable platform must be provided for the men to stand upon while raking the ore on the upper hearth; or, what is more commonly done, the level of the lower hearth is set inconveniently low, while the level of the upper hearth is still too high. It is the common practice, both in Europe and America, to build these furnaces in massives of four, with a common chimney in the center. This arrangement makes it difficult to install a proper dust chamber. In Kansas and Missouri, the double hearth reverberatory furnaces, which are there known as "flat kilns," have been displaced by shelf burners. A block of four, hand raked, reverberatory furnaces, two hearths high, once used in Europe, is shown in Figs. 28 to 30. The furnaces employed in Kansas were similar thereto.

Examples from Practice.—Results obtained in blende roasting in double

hearth reverberatory furnaces are presented in the following memoranda from various authorities and private notes:

Ammeberg, Sweden: The double hearth reverberatories are grouped four in a massive and are gas fired. Each single furnace roasts 3100 kg. of ore per 24 hours with the consumption of 545 kg. of coal, or 17·5%. The ore is burned down to 1·25% S. The massive of four furnaces is attended by five men per 12 hours, 10 men per 24 hours. The ore loses 26% weight in roasting.[1]

Cilli, Austria: Blende, poor in sulphur, is roasted in double hearth reverberatory furnaces, with hearths 7·5×2·5 m. (24·6×8·2 ft.=403 sq. ft. for the two). Three charges of 750 kg. each are roasted per 24 hours.[2] This is 4906 lb. per 24 hours and 12·3 lb. per sq. ft. of hearth.

Flône, Belgium: Furnaces of the same type and dimensions as those used at Oberhausen roasted 6283 lb. (2850 kg.) of ore per 24 hours, or 22·4 lb. per sq. ft. with consumption of 500 kg. of coal, or 17·5% of the weight of the raw ore.[3] These furnaces are now operated by two men per eight hours, instead of three men working 12 hour shifts as formerly, the number of men per 24 hours and the output remaining the same.

Hohenlohehütte, Upper Silesia: The double hearth reverberatory furnaces in use at these works have five working doors per hearth. A furnace holds three charges, viz. two on the upper hearth and one on the lower, each charge remaining for five hours before being moved forward; consequently the ore is 15 hours in the furnace. Each charge weighs 1000 kg. Every five hours a charge is dropped from the upper hearth to the lower, and a fresh one is put on the upper hearth. The capacity of the furnaces is therefore approximately 5000 kg. per 24 hours. The roast gases contain 1% SO_2 by volume. The sulphurous anhydride is absorbed by milk of lime, in a suitable tower, and the gases are finally dispersed through a chimney 100 m. high. The ore is burned down to 1% S.[4] Compare with the recent practice at the Silesiahütte, Upper Silesia. The consumption of coal and cost of labor in roasting at the Hohenlohehütte for 10 years is given in the following table,[5] wherein the quantities are stated in metric tons and the values in marks, one mark being equivalent to 23·8c. United States currency; the coal consumed was almost entirely slack (Staubkohlen), and the quantities reported include all that was used in the roasting plant, while the labor reported also includes the entire expense of the plant on that account, covering the handling of ore and coal as well as the furnace labor:

[1] P. Mahler, Ann. des Mines, VII, 512.
[2] Berg- u. Hüttenm. Ztg., 1894, No. 4.
[3] P. Mahler, loc. cit.
[4] Schnabel, Handbuch der Metallhüttenkunde, II, 48.
[5] Computed from statistics reported by the Oberschlesischen Berg- und Hüttenmännischen Verein.

Year.	No. of furnaces.	Ore roasted.		Coal burned.		Cost of labor. Marks.	
		Raw.	Product.	Total.	Per ton of raw ore.	Total.	Per ton of raw ore.
1891...	18	33,471	30,557	8,384	0·250	158,361	4·73
1892...	18	31,897	29,588	9,559	0·300	160,826	5·03
1893...	18	33,442	30,760	9,879	0·295	159,851	4·80
1894...	18	35,292	29,453	10,153	0·288	164,496	4·66
1895...	18	34.757	28,876	12.906	0·370	162,791	4·68
1896...	18	38,113	31,124	15,930	0·418	165,636	4·34
1897...	18	36,832	29,686	14,941	0·405	157,489	4·27
1898...	18	31,898	26,445	13,160	0·413	151,784	4·76
1899...	18	38,121	31,473	12,961	0·340	165,895	4·35
1900...	18	39,335	33,419	15,101	0·384	181,594	4·62

Malfidano, Sardinia: The plant comprises eight furnaces, which are grouped in massives of two each. One chimney serves the massive, but each furnace has its own fireplace. The hearths are 2·5×6 m. (8·2×19·7 ft. =323 sq. ft. for the two). They roast 2000 kg. (4409 lb.) per day, which is about 13·7 lb. per sq. ft. The desulphurized ore averages 60% Zn.[1] It is shipped to the smeltery at Noyelles-Godault, France, for distillation.

Oberhausen, Westphalia: The hearths are 6·5×2 m. (21·32×6·56 ft.= 280 sq. ft. for the two). The arch is 0·6 m. above the lower hearth, and 0·5 m. above the upper. Each furnace roasts 6614 lb. (3000 kg.) of ore per 24 hours, or 23·6 lb. per sq. ft. of hearth. The consumption of coal (containing 11% ash) is 1323 to 1543 lb. (600 to 700 kg.), i.e., 20 to 23·3% of the weight of the raw ore. The grate area is 1·31×6·56 ft.=8·6 sq. ft. (0·4×2 m.), the ratio of grate area to hearth area being 1 : 32·6. The coal was burned at the average rate of 6·4 to 7·5 lb. per sq. ft. of grate per hour. The raw ore assaying 25 to 26% S was burned down to between 0·57 and 0·83% S.[2]

Pittsburg, Kan.: Two furnaces are combined in a block. The lower hearth of each is 18 ft. 4 in.×8 ft.; the upper hearth, 23 ft. 4 in.×8 ft. The hearth area per block is therefore 666⅔ sq. ft. The arches over the hearths are 14 in. above the latter at the sides and 28 in. at the center. Each furnace is attended by one man, who works a 24-hour shift, which corresponds to four man-shifts per block per 24 hours. One man per 12 hours attends to 333⅓ sq. ft. of hearth. Each furnace yields 4000 to 4200 lb. of roasted ore per 24 hours, corresponding to 4600 and 4830 lb. of raw ore, and a hearth efficiency of 13·8 to 14·5 lb. per sq. ft. per 24 hours.

Reckehütte, Upper Silesia: In 1895 the double hearth reverberatory furnaces in use had hearths 6×2 m. (19·68×6·56 ft.=258 sq. ft. for the two).

[1] Zts. f. d. Berg- Hütten- u. Salinenwesen im Preuss. Staate, L, lii, 655.
[2] P. Mahler, Ann. des Mines, VII, 512.

Each hearth had six working doors. Each furnace roasted 7716 lb. (3500 kg.) of ore per 24 hours, or about 30 lb. per square foot of hearth.[1] The consumption of coal was 1000 kg., or 28·6% of the weight of the ore. A furnace was attended by one man per 12-hour shift, which corresponds to one man per 258 sq. ft. of hearth.

Silesiahütte, Upper Silesia: The hearths are 5·55×2·5 m. (17·53×8·2 ft. =288 sq. ft. for the two). Each furnace roasts 6614 lb. (3000 kg.) of ore, which is 23 lb. per sq. ft. of hearth per 24 hours, with the labor of two men. One man per shift rakes 288 sq. ft. of hearth.[2]

The double hearth reverberatory furnaces in use at the Silesiahütte (Lipine) in 1895 had hearths 5·6×2 m. (15·68×6·56 ft.=206 sq. ft. for the two). Each hearth had four or five working doors. Such a furnace roasted 11,243 lb. (5100 kg.) of ore per 24 hours, with the consumption of 1200 kg. of low grade coal;[1] this was 23·5% of the weight of the raw ore. The furnace crew comprised two men per 12 hours, or one man per 103 sq. ft. of hearth. The quantity of ore roasted was 54 lb. per sq. ft. of hearth, a very remarkable performance, although the hearth area per man is very small. The furnace contained three charges, of 650 kg. each, at one time.

Weir, Kan.: Furnaces of the same dimensions as at Pittsburg were used. A pair of furnaces used to roast 9400 to 10,000 lb. of ore per 24 hours, yielding 8000 to 8400 lb. of product, four drawings of 1000 to 1050 lb. being made from each furnace. The ore, assaying 28 to 30% S was burned down to between 0·70 and 1·50% S, varying according to its character. The rate of roasting was 14 to 15 lb. of raw ore per square foot of hearth per 24 hours. The number of men per furnace was the same as at Pittsburg. The consumption of coal (an inferior slack) was about one ton per ton of ore.

SUMMARY OF DOUBLE HEARTH REVERBERATORY FURNACES.

Place.	Hearth Area sq. ft.	Pounds ore roasted per 24 h.	Pounds per sq. ft. of hearth.	No. of man-shifts per 24 h.	Sq. ft. of hearth per man.	% Coal burned.	Pounds ore roasted per man-shift.
Ammeberg.....	280	6,834	24·4	2½	224	17·5	2,734
Ulli.....	403	4,960	12·3	..	....	....	.....
Flöse.....	280	6,283	22·4	..	....	17·5	.....
Hohenlohehütte..	....	11,023	....	..	....	35·0	.....
Oberhausen.....	280	6,614	23·6	..	....	21·5	.....
Pittsburg.....	333	4,830	14·5	2	333	....	2,415
Rerkehütte.....	258	7,716	30·0	2	258	28·6	3,858
Silesiahütte	288	6,614	23·0	2	288	....	3,307
....	206	11,243	54·0	4	103	23·5	2,811
Weir.....	333	5,000	15·0	2	333	....	2,500

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 49.
[2] Max Georgi, Berg- u. Hüttenm. Ztg., March 2, 1877.

SHELF BURNERS.—Shelf burners are simply multiple hearth reverberatories, which are built in single rows, double rows (back to back) or in blocks of four. Each furnace has usually four superimposed hearths, and sometimes five or six. The total length of hearth is limited by the same factors which limit the length of a two-hearth furnace, or a long single hearth reverberatory. Instead of the ore being worked forward by tools inserted through side doors, however, the shelf burners have the working doors in the end, wherefore the operator draws the ore toward him on one hearth and pushes it away from him on the next one below. The length

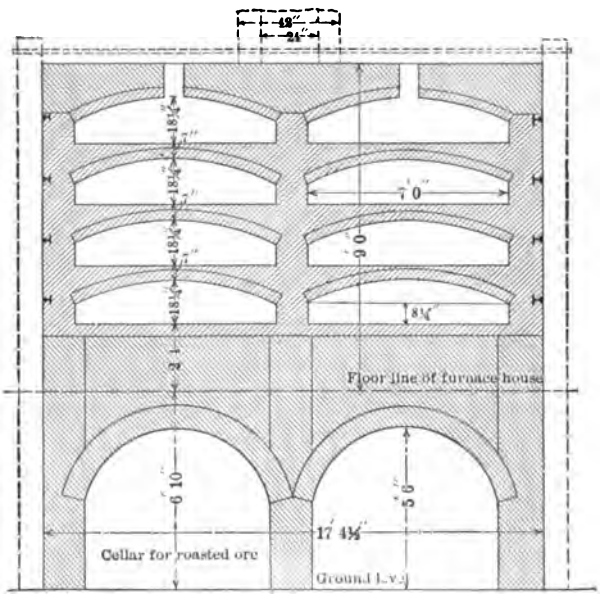


FIG. 31. TYPE OF SHELF KILN USED IN KANSAS.
Transverse vertical section.

of an individual hearth is limited only by that of the tools which the operator can conveniently handle. This is probably about 9 ft. The hearths of shelf burners are generally made, however, about 8 ft. long and 8 ft. wide. Common dimensions are 7×7 ft., 7×8 ft., 8×8 ft. and 8×9 ft.

Shelf burners are largely used for blende roasting in Kansas and Missouri, although they have been recently displaced to a considerable extent by mechanical furnaces. A typical kiln of the type employed there is shown in the accompanying engravings. These kilns are generally built in blocks of four, i.e., the block is divided into four sections by vertical walls.

One wall divides the block into two sections, which are further subdivided by the fireplaces, which thus separate two opposite sections. Each section has four superimposed hearths, but kilns with five or six hearths are frequently used. The products of combustion unite in a flue, over the arch of the fireplace, which communicates with a central chimney; sometimes, however, the longitudinal transverse wall is carried up through this flue and there are two chimneys, one for each fireplace.

In Kansas and Missouri these kilns are commonly built high above the ground, so that the roasted ore can be dropped into vaults on the ground

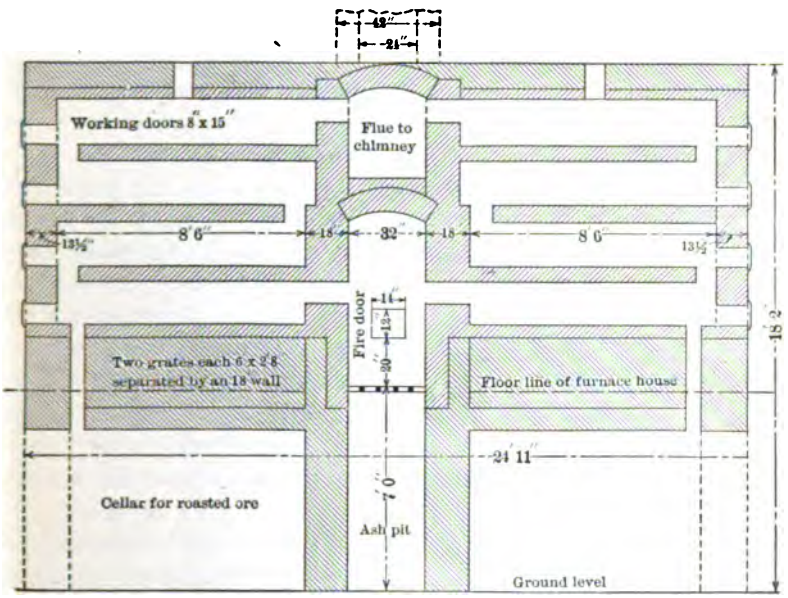


FIG. 32. TYPE OF SHELF KILN USED IN KANSAS.
Longitudinal vertical section.

level, from which it is trammed directly to the distillation furnaces. There is also a deep ash pit which gives easy access to the grates. The latter consist of a few wrought iron bars, laid loosely on transverse rests, which support a thick bed of clinkers on which slack coal is burned, this constituting a crude system of semi-gas firing, the same as is practised with the distillation furnaces of the district (q. v.). At Iola, similar kilns are used with natural gas firing. In Indiana, kilns of the same type are used, but they are built lower, without the high vaults for roasted ore under the hearths, and at some works they are arranged in a single row instead of in

blocks of four or double rows. As at Iola, Kan., all the roasting kilns in Indiana are fired with natural gas.

The doors through which the ore on the hearths of these kilns is raked are placed in the ends. They are commonly iron castings hinged on a rectangular frame set in the brickwork in the usual manner. A common size is 8×15 in. Sometimes there are two doors per hearth; sometimes there are two doors to the hearth on which the ore is pushed away from the operator and only one to the hearth on which it is pulled toward him. The lower hearths are reached from the plank floor of the house surrounding the furnace; to enable the operator to reach the upper hearths a movable platform is pushed up in front of the furnace. The brick walls are supported by horizontal buckstaves, only on two opposite sides of the block, which are connected at the ends by tie rods passing just over each row of working doors. These horizontal beams form a solid support for the skewbacks of the arches. Outside of the horizontal buckstaves, vertical buckstaves are set and tied across the top of the furnace in the usual manner. In those kilns which have alternately two and one working doors per hearth vertical buckstaves on the side from which the hearths are raked would be in the way, except at the corners; there is no good reason why they should not be used on those sides of the furnaces which have two doors per hearth. However, the method of staying employed in Kansas and Missouri gives apparently results which are good enough, since the repairs required for these kilns are relatively very small, and their life is long. The repairs most frequently required are relining the fire boxes, which has to be done at intervals of one to two years; repairs on the hearths are needed less frequently.

A four-hearth kiln like that shown in Figs. 31 and 32 has an effective hearth area of about 840 sq. ft. The ore in roasting is allowed to lie on the hearth in a bed about 2·5 in. thick. With two men per shift about five tons (10,000 lb.) of raw ore will be put through in 24 hours, the tenor of sulphur in the roasted ore being less than 1%. This is about 12 lb. per sq. ft. of hearth area. There are two grates, each 6×2 ft. 8 in., giving a total grate area of 32 sq. ft. and a ratio of grate to hearth of 1 to 26 approximately. The consumption of coal varies much according to the quality used. It appears to range from 50 to 100% of the weight of the raw ore; in most cases, probably, nearer the latter figure than the former. An inferior slack is generally employed. It is commonly obtained, however, at so low a price that the large consumption per ton of ore is not a very serious charge on the cost of roasting. In this respect the conditions existing in the United States are very different from those which prevail in Europe.

The furnace crew consists of two men per shift, each man having one grate and eight hearths to attend to. It is a common practice to pay the men according to the product turned out, a limit being set on the percentage of sulphur that it may contain. When this is done the men will usually work 24-hour shifts, day on and day off. Contracts are made at 7c.@8c. per 100 lb.=$1·40@$1·60 per ton of roasted ore, or $1·19@$1·36 per ton of raw ore; for the higher figure the men frequently carry the ore from the storage bins to the roasting furnaces, but the roasted ore is removed at the company's expense. At 7·5c. per 100 lb. a man will earn about $3·75 per 24 hours=$1·875 per 12 hours. In Indiana roasting is contracted at 5c. per 100 lb. of product, the kilns being heated by natural gas, and the men earn $1.75@$2 per 12 hours at that figure. The same rate is paid in the natural gas smelteries of Kansas. This makes the cost for labor in roasting a ton of raw ore about 85c., which is certainly a very low figure considering the scale of wages that prevails.

Perhaps the most general construction of the shelf burners in the natural gas districts of Kansas is eight double furnaces in a block, back to back, four in a row. By double furnace is meant here one furnace of three hearths superimposed upon another of the same number, the two together being six hearths high. This curious practice has been developed in the gas fields alone, the ability to pipe the gas in almost any manner desired enabling a construction which would hardly be feasible if the furnace had to be heated from a fireplace. The two furnaces, one over the other, are quite distinct in their operation. Indicating the hearths of the upper by A1, A2, A3, and those of the lower by B1, B2, B3, the raw ore is charged to B1 by a chute in the rear wall and the roasted ore is discharged from A3 by a chute in the front wall. Hearths A3 and B3 have one working door, on each side of which a gas burner is introduced. The products of combustion escape through flues in the rear wall to a main flue on top of the block, which may have an independent chimney, or the flues of several blocks may lead to one large chimney. Such a block of eight double furnaces, or 16 single ones, generally roasts about 18 to 20 tons of ore per 24 hours, delivering 15 to 17 tons of product, the furnace crew comprising four men per shift who are paid the customary 5c. per 100 lb. of product. The blocks are built high above the ground in order to permit the installation of the necessary pockets for receipt of the roasted ore, which is drawn therefrom into a car, and the multiplication of hearths causes these massive structures to tower to great height. The placing of one furnace above another saves something in the first cost of the foundation and in the furnace building, which does not have to cover so large an area as otherwise, while as to inconvenience

in operation there is probably no difference, since the lower furnace itself is so high as to require movable platforms for the men to stand on. Occasionally the furnaces at Iola are built with more than four in a row, and that is also the case in Indiana, but in the latter State they are not the elevated structures they are in Kansas.

Examples from Practice.—A works at Pittsburg, Kan., had shelf burners grouped in blocks of four burners each. The burners were four hearths high, wherefore there were 16 hearths per block. The hearths were 6 ft. 9 in. wide and 9 ft. 6 in. long. The arches over the hearths were 21 in. above the latter at the center and 9 in. at the sides. The total hearth area per block was 1026 sq. ft. The roasting was done by two men per block per 24 hours, i.e., there were four man-shifts. Three charges were dropped in and drawn out per burner per 24 hours, i.e., each charge remained eight hours on a hearth, and was 32 hours in passing through the furnace. The single charges weighed about 940 lb. and yielded 800 lb. of roasted product. The quantity of ore roasted per block per 24 hours was 11,280 lb., the product being 9600 lb. The hearth efficiency was 11 lb. per sq. ft., one man having to attend to 512 sq. ft. of hearth. The ore was burned down to 1 to 1·30% S. The consumption of coal was approximately five tons per 24 hours, or about 90% of the weight of the ore roasted.

A block of eight burners, each of three hearths, or a total of 24 hearths, as above described, at a certain works at Iola, Kan., has hearths 9×9 ft., giving a total hearth area of 1944 sq. ft. The arch over each hearth is 20 in. above the latter at the center and 10 in. at the sides. The ore is crushed to between 1·5 and 2 mm. size and is charged in lots of 2500 lb.,[1] a charge being dropped in and drawn out every 12 hours; consequently the ore is 36 hours in passing through the furnace. The ore lies on the hearth in a bed about 3 in. thick. The loss of weight in roasting is 12 to 13%, a charge of 2500 lb. yielding about 2200 lb. of roasted product. The ore is burned down to 1 to 1·5% S. The capacity of such a block of eight burners is $2 \times 8 \times 2500 = 40{,}000$ lb. of raw ore per 24 hours. The furnace crew comprises four men, who work 24-hour shifts, corresponding to eight-man shifts of 12 hours.[2] Each man attends to six hearths, or 486 sq. ft. The hearth efficiency is 20·5 lb. of ore per 24 hours per square foot. In drawing 35,200 lb. of roasted ore at 5c. per 100 lb. the men receive $2·20 per shift of 12 hours; in drawing 32,000 lb. they would earn $2 per shift.

[1] In very hot weather, the charges are lighter, say 2300 lb. instead of 2500. This variation is common in the Kansas smelting practice. In mid-summer the temperature is frequently 100° F. in the shade.

[2] Otto Rissmann, private communication.

Reverberatory Furnaces Raked Mechanically.

Mechanically raked reverberatory furnaces are subdivided into those with stationary hearths and those with movable hearths; in the former the rakes are movable; in the latter they are stationary. In both kinds the principles of combustion, air supply, etc., are essentially the same as in hand raked reverberatories.

FURNACES WITH STATIONARY HEARTHS.—Mechanical furnaces with stationary hearths are by far more important as a type than are those with movable hearths. They have been built with long rectangular hearths, with circular hearths, and with annular hearths. Circular hearths are necessarily of rather limited diameter, but annular and rectangular hearths can be made of great dimensions. Because of the ability to place auxiliary fireplaces along the sides, or around the periphery of these furnaces, the length of hearth can be made as much as desired, inability to heat as in the case of the hand raked reverberatory being no limitation.

Brown Furnace.—The Brown furnaces were the first in which the difficulties connected with the O'Harra, which was the prototype of this class of mechanical furnaces, were successfully overcome.[1] It was originally built in the form of a broken ring, known as the "horseshoe furnace," next in an elliptical form, and finally in a rectilinear form, known as the "straight line" furnace. Each of these forms is built now according to the choice of purchaser, but only the horseshoe furnace has been employed so far in blende roasting in the United States, though a straight line muffle furnace has been installed in Upper Silesia. The right to all of the Brown furnaces for blende roasting in Indiana, Illinois, Kansas, Missouri and Arkansas is controlled by the Edgar Zinc Co. in conjunction with the Collinsville Zinc Co. They are built by the Allis-Chalmers Co.

In its most improved form the Brown straight line furnace is a single hearth reverberatory with an interior slotted wall on each side of the hearth, covered with a low arch (see Fig. 33). In the conduits formed by the slotted walls and the exterior walls of the furnace, are placed rails upon which run trucks supporting a stirrer arm, projecting through the slots. Fastened to that portion of the arm which extends over the hearth are stirrer shoes, resembling miniature plowshares, which dip down into the ore, lying in a thin layer on the hearth. The stirrer carriages (there are never less than

[1] The essential improvements which were embodied in the Brown furnace were the recessed chambers in which the mechanism of the rake carriage is protected from the heat and the continuous slot through which the rake arm passes. The patents covering those features have lately been sustained after a long litigation.

two) are moved through the hearth chamber by two endless steel link chains, which pass over sprocket wheels located at both ends of the furnace outside of the roasting hearth. After passing through the hearth chamber each carriage returns over the top of the arch on tracks similar to those placed in the conduits, and having reached the end of the arch it passes downward again into the hearth chamber. There are sheet iron flap doors, hinged at the top, at each end of the hearth to keep out the cold air; these remain closed except when lifted by a stirrer carriage passing in and out.

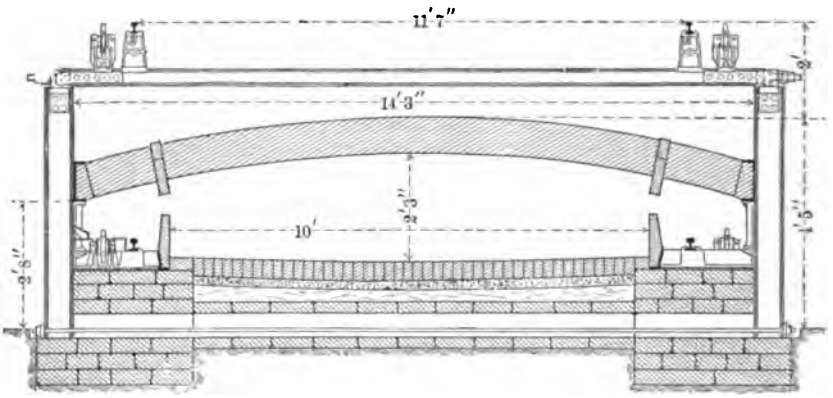


FIG. 33. BROWN STRAIGHT FURNACE.
Transverse section through hearth.
Scale, ¼ in. = 1 ft.

By this construction the operating mechanism is protected from the furnace heat, and the carriages, by passing over the top of the furnace into the open air, have time to cool and the stirrers do not become overheated.

The slotted walls are formed by fire clay tilings which project upward from the hearth and by fire brick which project downward from the arch. The skewbacks of the arch are steel channel beams supported on columns 12 in. high, the spaces between the columns being 3 ft. 6 in. long. These openings, which extend the entire length of the hearth on both sides, are closed, ordinarily by sheet iron doors lined with asbestos. This construction permits of ready access to the hearth at any point for repairs, etc., without having to tear down the brick construction to get to the interior of the furnace. From the ground to the hearth the furnace may be conveniently built of uncut stone, and the rest of the furnace constructed of brick, the structure being bound together by steel I beams.

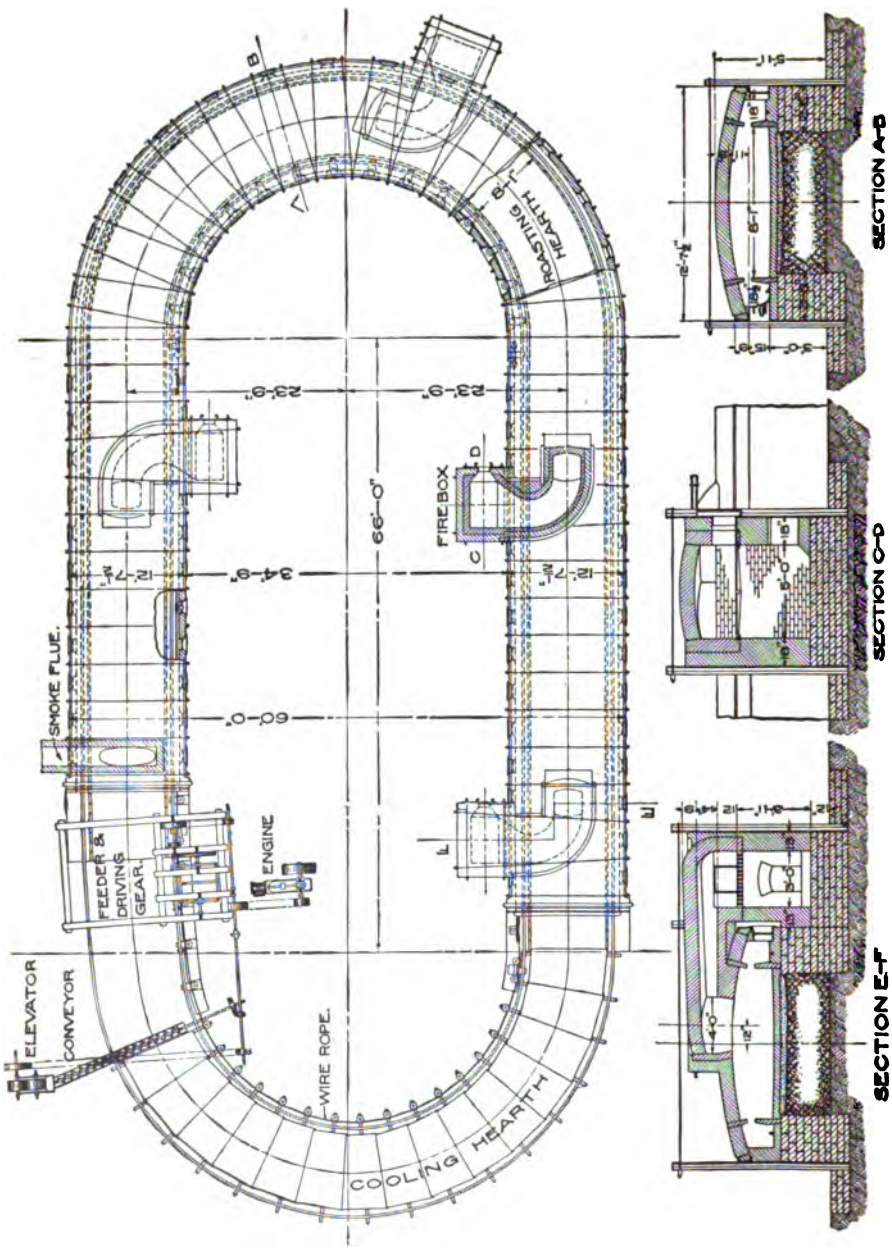
By having the stirrers pass over the top of the furnace the brick structure is built low and the solid hearth conserves heat against radiation. Fire

boxes, in number and position to suit the length of the furnace and the character of the ore, are provided; they are arranged in pairs opposite each other in the usual construction. When subsequent operations render it necessary to cool the roasted ore rapidly, the top of the furnace is converted into a cooling floor by covering it with thin steel plates having upturned edges, the construction being that of a shallow tray the length of the arch and the width of the roasting hearth; special double arm stirrers are provided with which to raise the ore from the roasting hearth to the cooling pan along which it is gradually advanced, in the same manner as on the roasting hearth, to the cool end of the furnace, where it may be carried away by a conveyor or other device.

When the Brown furnace is made other than straight, the stirrer carriages are drawn by an endless steel wire rope which runs in the inner conduit and is supported on small horizontal grooved wheels placed sufficiently near each other to cause the rope to conform to the curve of the furnace; the rope is driven and kept taut by passing it around a system of wheels driven by a small slide valve engine provided for that purpose. The rope is connected with the carriages by means of a small grip similar to that used on cable cars, the construction being such that as soon as a carriage emerges from the hearth into the open air it is unclutched automatically and remains there cooling off until the carriage behind it emerges and strikes it, pushing it forward a short distance to a point where it becomes again clutched to the rope and travels forward while the second carriage becomes unclutched and remains stationary.

It has been the experience of the builders of these furnaces that in the straight furnace the best width for the actual roasting hearth is 10 ft., and for the furnaces which have a curved hearth, 8 ft. The straight furnace may vary from 60 ft. to 180 ft. in length, and the diameter of the round ones may be from 50 ft. upward. In places where economy in floor space need not be considered, the round or the elliptical type may be used, being simpler in construction and costing less per square foot of hearth area than the straight furnace, but where space is limited the straight furnace should be used. The largest Brown furnaces of the round or "horseshoe" type are 119 ft. in diameter and have 2422 sq. ft. of effective roasting hearth. These were installed at the works of the Edgar Zinc Co. at Cherryvale, Kan. The engraving Fig. 46 shows these furnaces in course of construction.

The arrangement of a Brown elliptical furnace with a cooling hearth is shown in the accompanying engraving. The roasting hearth is 180×8 ft. and the cooling hearth 78×8 ft. In the construction of such a furnace there is required 50,000 lb. of iron work, 90,000 red brick, 8100 fire brick, 12 tons



FIGS. 34 TO 37. BROWN ELLIPTICAL FURNACE.

of fire clay, 250 bushels of lime, and 100 cu. yds. of sand. The foundation requires 400 perches (10,000 cu. ft.) of stone work. The erection is accomplished with about 250 days of common labor, 250 of masons' and 20 of smiths' and machinists' labor.[1]

The details of the construction of the circular or "horseshoe" furnace is shown in the accompanying engravings, which are reproduced from Professor Hofman's treatise, *Metallurgy of Lead.* In the center there is an engine *a* of 5 to 8 h. p. (although only 2 h. p. are required), which drives the mechanical feeder *b* and the cable *c* running over sheaves *d.* Opposite each sheave is a door *e,* which gives access to the sheave for oiling and repairing and admits air for cooling the bearings. The outside doors *f* are 4 ft. apart, which is just enough to give access to the hearth when the outer wall *g* between the doors has been removed and the tiling *h* taken out. The flues from the exterior fireplaces are turned in the direction of the draught, so that the flame may spread uniformly under the roof and not strike the ore. The gases and fumes are carried off by the chimney *i.* The ore is fed at *b* by an automatic feeder, which delivers with every passage of a carriage a weighed quantity on the hearth outside of the swinging door. The ore enters the furnace through the swinging door *j,* travels over the hearth, leaves it through the other door *k,* and is discharged into wheelbarrows or trucks placed at *l.* The stirring carriage *m* after leaving the furnace strikes a second one at rest (not shown in the engraving) in the open space and pushes it along until its grip-lever comes in contact with a fixed stop and is forced down, when it grips the cable which draws the carriage into the furnace. At the same time the heated carriage is automatically released, comes to a stop and cools in the open space for from one to three minutes, which is the time required for a carriage to make the circuit. In this manner the two carriages are used alternately at intervals of from one to three minutes and are not sufficiently heated to be much affected by their passage through the furnace.

On the side walls of the furnace, built to a level with the hearth, are placed the heavy cast iron frames of the inner and outer doors *e* and *f*; on the frames rest 9 in. channel beams *n*, secured by the buckstays *o* and the tie rods *p*. Thus the roof *q* is supported independently of the brickwork between the doors, which can be taken out if necessary. The sheaves *d* are supported in the doors outside the inner passage, which prevents them from being much heated. The tiling *h*, forming the lower part of the slotted wall, is set in the brickwork of the hearth and can be readily removed without disturbing the roof or cooling down the furnace. The tile *s*, form-

[1] John E. Rothwell, in The Mineral Industry, V, 270.

ing the upper part of the wall, is built into the roof and special precautions must be taken to secure it firmly. The inner passage shows a T rail for the inner double-flanged wheels of the stirring carriage, while in the outer passage there is a smooth rail for the outer flat-tread wheels, giving room

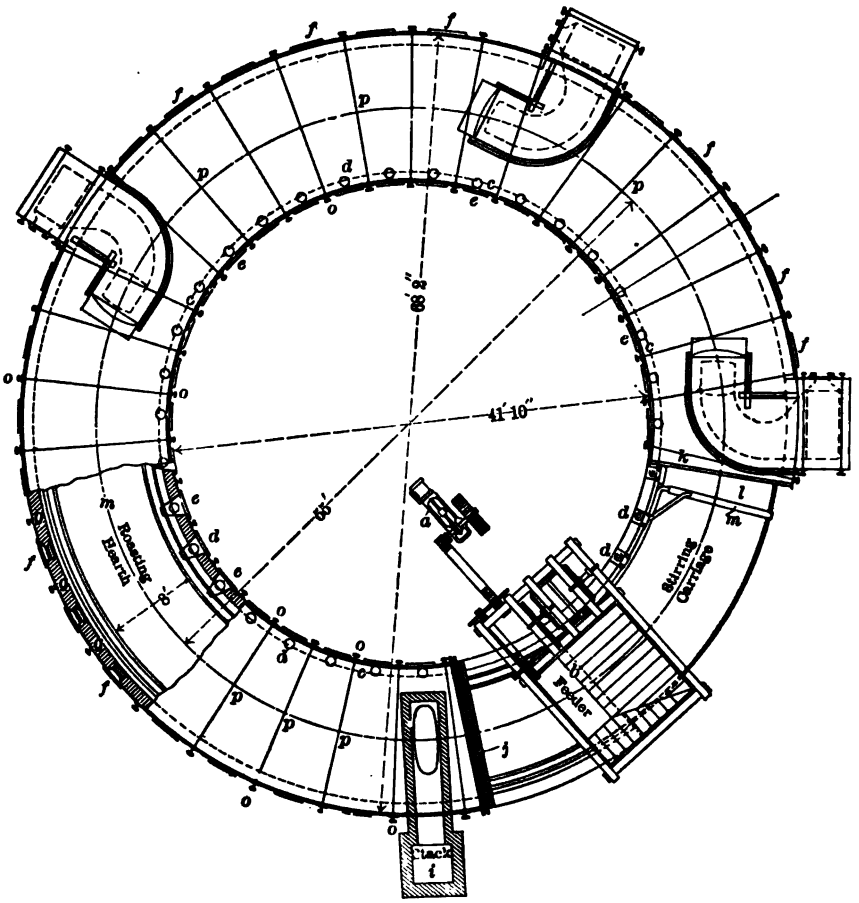


FIG. 38. BROWN HORSESHOE FURNACE.
Plan.

for expansion and contraction. At some works the furnace is inclosed by ribbed cast iron skewbacks at the level of the tracks in order to keep the latter rigidly in place. The first fire box is placed about 60 ft. from the feed end of the furnace, the other two are 35 and 40 ft. apart.

The stirring carriage was originally fitted with plows, made of oblong plates of steel placed obliquely to the arm reaching across the hearth. Such a plow wore off from 1·25 to 1·5 in. in about a fortnight, when it was turned, and after another two weeks was discarded. This form of plow had the disadvantage that the front point wore more rapidly than the rest of the blade, with the consequence that the ore would not be stirred uniformly; in recently built furnaces the uneven wear has been diminished by riveting to the blade a white-iron shoe, which wears very slowly. When crusts form on the hearth, they are sometimes plowed up, say once in five days, by a curved steel bar attached to a reserve carriage. The bar is shifted about 5 in. after each

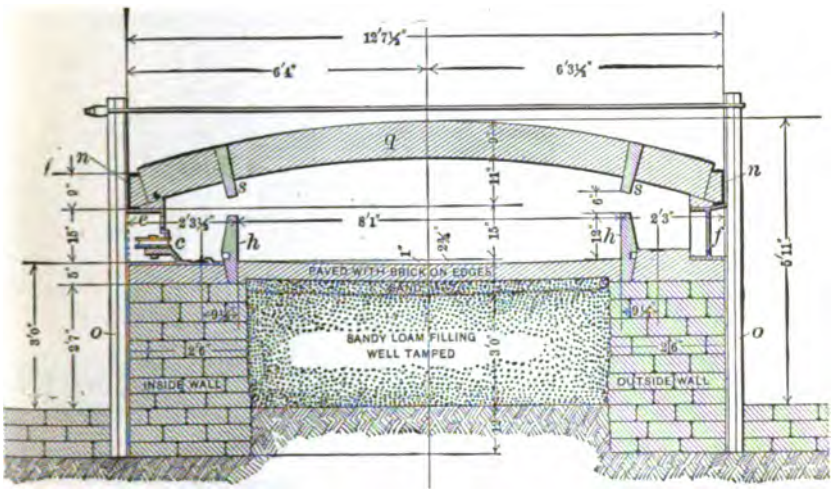


FIG. 39. BROWN HORSESHOE FURNACE.
Transverse section through roasting hearth.

circuit and the crust broken up in about an hour. Care must, of course, be taken not to allow the crust to become too thick, since when that is the case, plowing up strains the machinery and becomes almost impossible. Another way is to have a separate frame with steel teeth, 1 in. square, reaching beneath the ore and run it all the time, thus preventing any crust at all from forming. The improved rabble *t* (Fig. 43) is plow-shaped and of cast iron; it wears on the bottom, but the cutting edge always remains pointed. The plow is keyed to a cast steel rabble arm *u*, attached to the stirrer arm *v* reaching across the hearth. The arm can be lowered as the plow wears off. When a crust forms the arms are lowered one at a time so as to cut just under it into the loose material lying on the brick hearth.

The rabble arms are placed about 12 in. apart on each carriage (Fig. 44), and are mismatched, so that with three carriages the whole hearth surface is covered. One of the carriages has side stirrers *w*, to prevent any ore from accumulating there. A carriage (Figs. 44 and 45) consists of an L-shaped frame with one flat-tread wheel *x*, running in the outer, and two double-flanged wheels *y* running in the inner passage. The front inner wheel has the cable grip *z*.

The life of a cable is six months and less according to the strength of the splice and the strain it has to bear. The "Flattened Strand Rope" lately introduced, presenting a much greater wearing surface, is an improvement on the common round cable. The wheels of the carriage, formerly of cast iron and wearing out in two months, are now made of cast

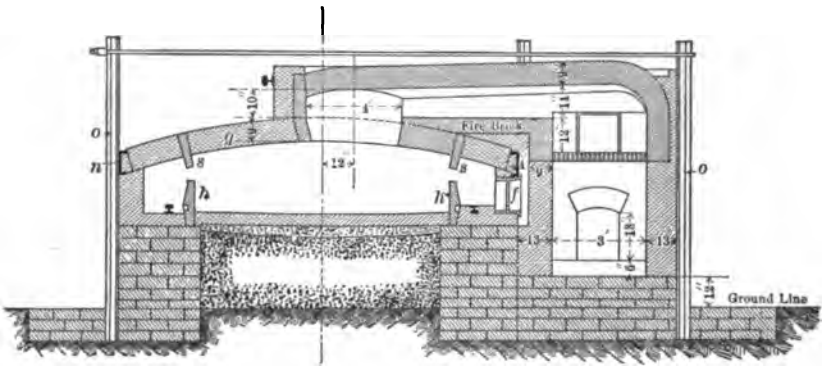
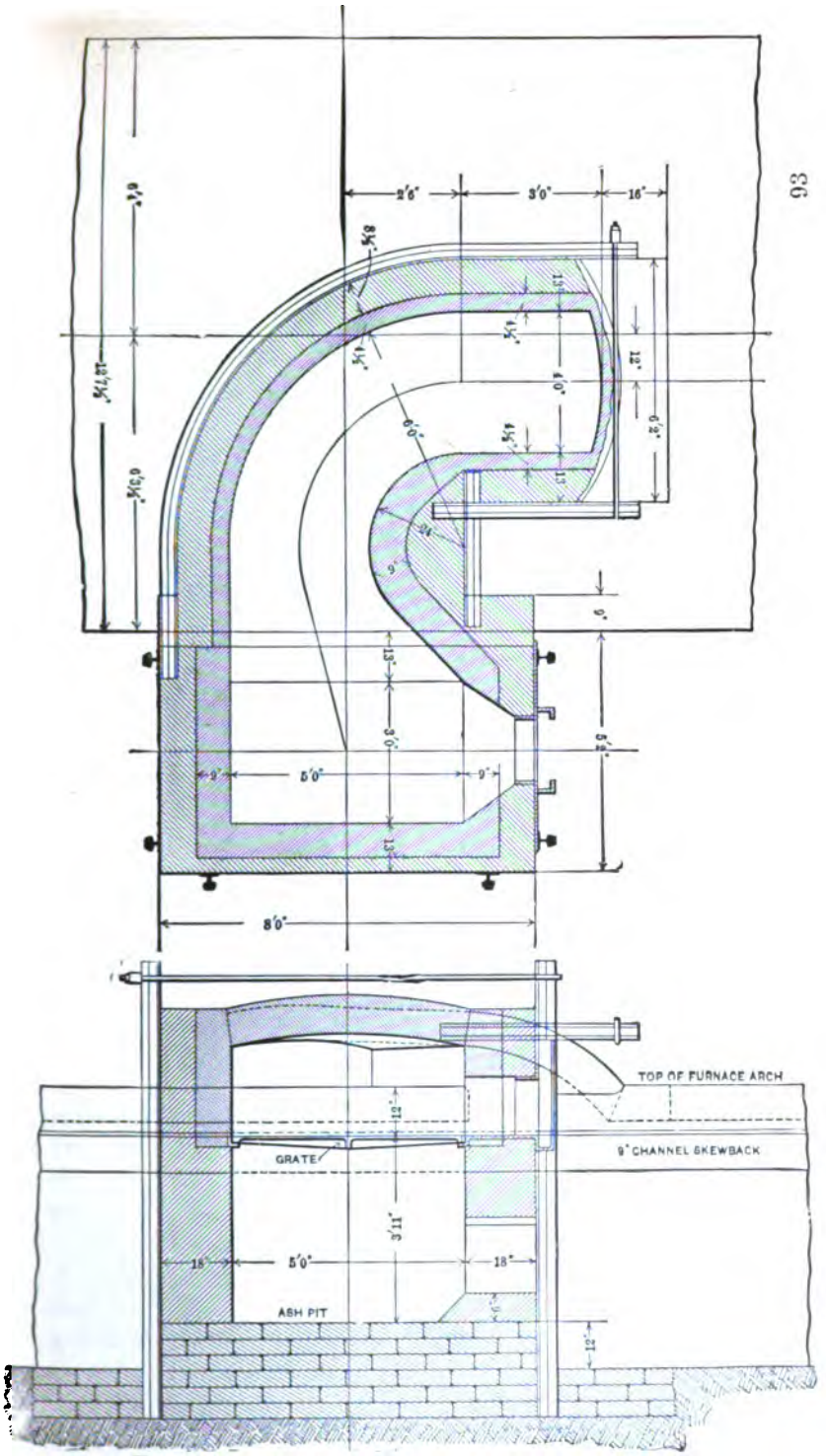


FIG. 40. BROWN HORSESHOE FURNACE.
Transverse section through fire box and roasting hearth.

steel and last about four months. The cast iron plows last from four to five times as long as those made of steel plate.

The furnace just described has a hearth 135 ft. long and 8 ft. wide, giving an area of 1080 sq. ft. Its inner diameter is 41 ft. 10 in. and its outer diameter 68 ft. 2 in. The fireplaces project 5 ft. 2 in.; therefore a floor space of 6162 sq. ft. is required. The total height of the furnace is 6 ft. The grates of the fire boxes are 3×5 ft., making the ratio between the total grate area and the hearth area 1:24. The materials required for such a furnace are: Iron work, not including buckstays, 35,000 lb.; buckstays, 20,000 lb.; iron chimney, 42 in. diameter and 80 ft. high, including base plate, 5200 lb.; red brick, 60,000; standard fire brick, 4500; special fire brick and tiling, 12·5 tons. The cost of the iron work f. o. b. Chicago in 1898 was $3500 and the total cost of the furnace erected, $7000.



FIGS. 41 AND 42. BROWN HORSESHOE FURNACE.

Horisontal and vertical sections through fire box.

The Brown horseshoe furnaces employed for roasting blende at Collinsville, Ill., are 55 ft. in diameter, measuring to the center line of the hearth, and have an effective roasting area of 8 ft. wide and 135 ft. long, or 1080 sq. ft. There are four fire boxes, arranged for burning slack coal, the combined grate area being 48 sq. ft. or 1 sq. ft. per 22·5 sq. ft. of hearth area. The stirring mechanism consists of two carriages drawn by a continuously running rope, the rabbles on each being set alternately. The ore to be roasted, crushed to about 12-mesh size, averages 58% Zn, 28 to 30% S and 1% Pb, besides a small percentage of iron. The ore is fed on an extension of the furnace hearth, outside of the end doors, the feed being regulated by

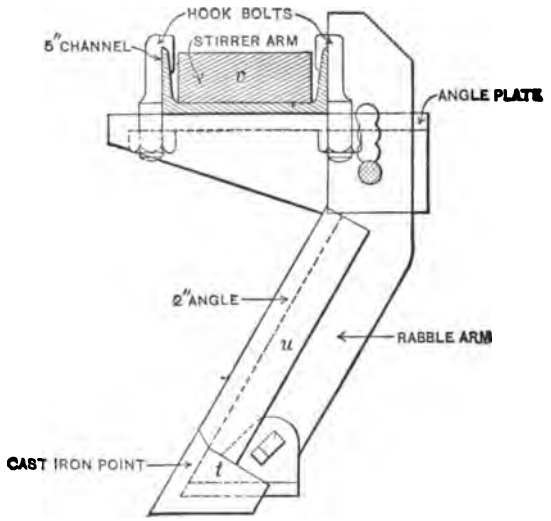
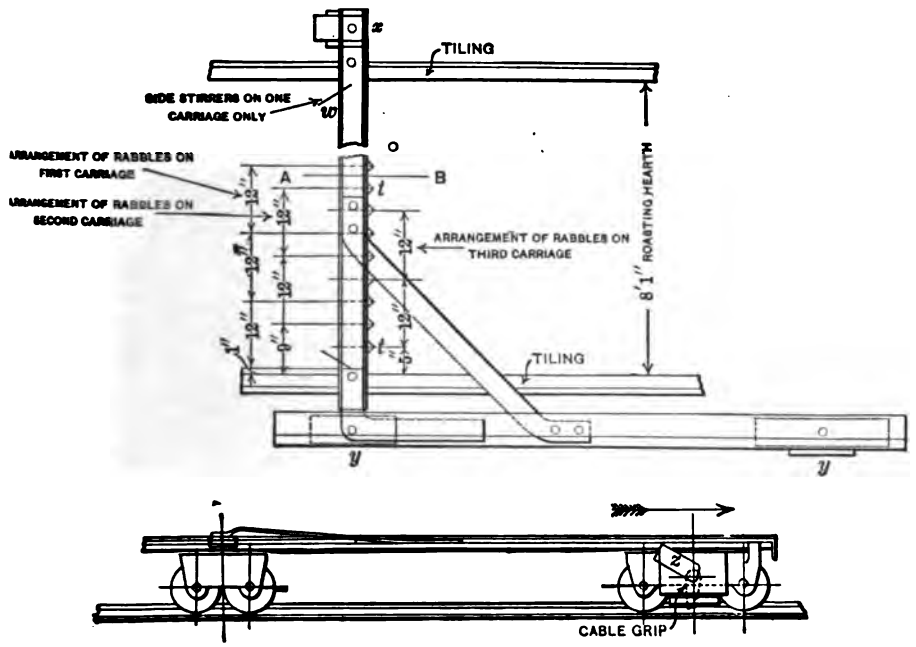


FIG. 43. BROWN HORSESHOE FURNACE.
Details of rabble arm; section on line A B of Fig. 44.

means of a weighted lever, ratchet and pawl. A predetermined quantity of ore is fed with the passing of each carriage, after which the weighted lever falls, throwing the pawl out of gear and stopping the feed. The feed mechanism is put in motion by the carriage. The ore is fed through a canvas hose, the bottom of which is fitted with a sheet metal thimble about 12 in. wide and the full width of the hearth in length. This hose is lifted by means of a lever engaging the carriage arm, so as to allow the carriage to pass, and falls behind the carriage just before the feed commences, the metal bottom cutting into the bed of ore on the hearth and thereby forming a dust-tight joint. All the dust due to the dropping of dry ore is retained

in the hose and settled before the next charge, making practically a dustless feed.

The ore enters the furnace chamber immediately under the flue to the chimney. The temperature of the escaping gases at that point is about 427° C. (800° F.), it having been found best to start with a high initial temperature. The ore attains red heat after traveling about 10 ft. in the furnace,



FIGS. 44 AND 45. BROWN HORSESHOE FURNACE.

Plan and elevation of stirring carriage.

which occupies about 1·5 hour. The progress in desulphurization and the temperatures attained in this furnace were stated in Chapter III. The quantity of raw ore roasted in 24 hours is 11 to 12 tons, the consumption of coal being 9 to 10 tons per 24 hours, i.e., about 83⅓% of the weight of the raw ore. The coal is slack of inferior quality. One man per shift of 12 hours, paid $2·50, attends to a furnace.[1] The cost of power, repairs and renewals, and miscellaneous supplies is said to come to about 25c. per ton of ore.[2] The cables last an average of eight months, and the cast iron rabble

[1] In the more recent practice the furnacemen have worked eight-hour shifts, being paid $1·60 therefor.

[2] The care of four furnaces occupies a smith most of his time.

shoes about four months. After four years' use the tracks showed no material deterioration. At the Collinsville works there are four of these furnaces, with one chimney for each pair.

At the Glendale works of the Edgar Zinc Co. at Carondelet, near St. Louis, a Brown horseshoe furnace of the same dimensions as those at Collinsville puts through 11 to 12·5 tons of ore assaying 30% S per 24 hours, which is roasted down to 0·5% to 0·7% S with a coal consumption of 66⅔% of the weight of the raw ore. The chimney which serves the Glendale furnace is 36×36 in. at the top and 90 ft. high. The furnace crew per 12 hours consists of one roaster and one trammer. At the Cherryvale works of the Edgar Zinc Co. there are three horseshoe furnaces, each of 2422 sq. ft. hearth area, which are heated by natural gas.

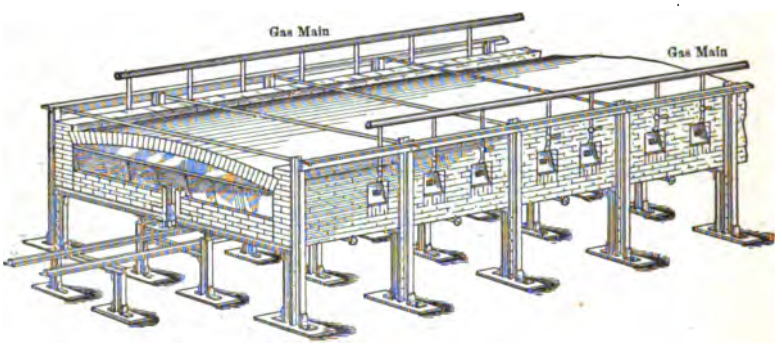


FIG. 47. CAPPEAU FURNACE.

Cappeau Furnace.—Joseph P. Cappeau, general manager of the Lanyon Zinc Co., recently patented a mechanical roasting furnace, an example of which is in use at one of the Iola works of that company. This furnace is more or less a modification of the Ropp. The chief differences are that the roasting chamber is like a long box supported on posts, leaving the lower side of the hearth exposed to the air, and the admission of air through the longitudinal slot is regulated by means of adjustable gates, which are pivoted so as to be pushed sidewise in order to permit the passage of the rake arm, afterward dropping back to their normal position according to the adjustment. The general appearance of the furnace is shown in the accompanying engraving. The patent specifications claim: "A roasting furnace comprising a hearth supported on standards to permit free circulation of air beneath the floor of said hearth, an opening in said floor for permitting

FIG. 46. BROWN HORSESHOE FURNACE IN COURSE OF CONSTRUCTION AT CHERRYVALE, KAN.

the passage of air into the hearth, and adjustable means for controlling the passage of air through said opening."[1]

Davis Furnace.—This is a long, one hearth reverberatory, heated by lateral fireplaces. On each side of the hearth there are pockets in which spur wheels are set on short horizontal shafts, the latter being parallel with transverse lines through the furnace. Outside of the furnace, a bevel gear on the end of each of the short shafts meshes with a pinion on a main driving shaft, which is parallel with the longitudinal axis of the furnace. The rabble consists of a rake with adjustable teeth, which is fixed between long side bars at each end. These side bars have on their lower sides long racks, which engage with the spur wheels above referred to. Consequently, when the spur wheels are turned by means of the driving shaft, outside of the furnace, the rabble is caused to travel forward. At the end of the forward movement, the direction of revolution of the driving shaft is automatically reversed and the rabble is thereby caused to move backward, the rake blades being then turned to a horizontal position. The rabble remains always in the furnace. It consists of the two side bars, which are open at the top, like troughs, and three hollow tranverse bars, the middle one bearing the rakes. The side bars and the transverse bars are kept full of water, the loss by evaporation being automatically made good after every trip through the fire. The rakes are set so as to insure a thorough stirring of the ore and the rabble is so designed as to prevent any pinching of the racks with the spur wheels upon which they travel.

This furnace, which is built by the F. M. Davis Iron Works Co., of Denver, Col., is to be given a trial for blende roasting at the works of the Lanyon Bros. Spelter Co. at Neodesha, Kan. The hearth dimensions are 150 ft. × 12 ft., giving an area of 1800 sq. ft. In the construction of the furnace there will be used 65,000 lb. of iron work, 100,000 red brick, 42,000 fire brick, 55 bbl. of cement, 150 bushels of lime, 43 cu. yd. of sand and 20,000 lb. of fire clay.

Hall Furnace.—This is a recently patented furnace[2] which may be considered as a modification of the Wethey, the stirring carriage traveling on rails outside of the furnace, but the slot through which the horizontal arm passes is water-sealed, instead of being closed by tripping gates. On each side of the hearth there is an iron water trough. The skewbacks of the arch are carried by cast iron angle beams, bolted to the buckstaves, and from the

[1] United States patent No. 691,112, Jan. 14, 1902. It is reported that all of the Ropp furnaces of the Lanyon Zinc Co. are being altered to conform to this new design, which is obviously intended to evade the Brown patents, of which the Ropp has been declared to be an infringement.

[2] Benjamin Hall, United States patent No. 677,510, July 2, 1901.

bottom of the latter there is a vertical flange which dips into the water in the trough. The ends of the horizontal rake arm are bent down in U shape, so as to pass through the trough, under the downward projecting flange. The roasting chamber is thus sealed perfectly on the sides.

Keller Furnace.—This furnace is a modification of the Spence. It consists of two five-hearth kilns, which are built 8 ft. apart, measuring between opposite walls. The hearths are 21 ft. 5 in. long and 6 ft. 3 in. wide. The hearths are 3 ft. apart vertically. They are built of red brick, as indeed the whole

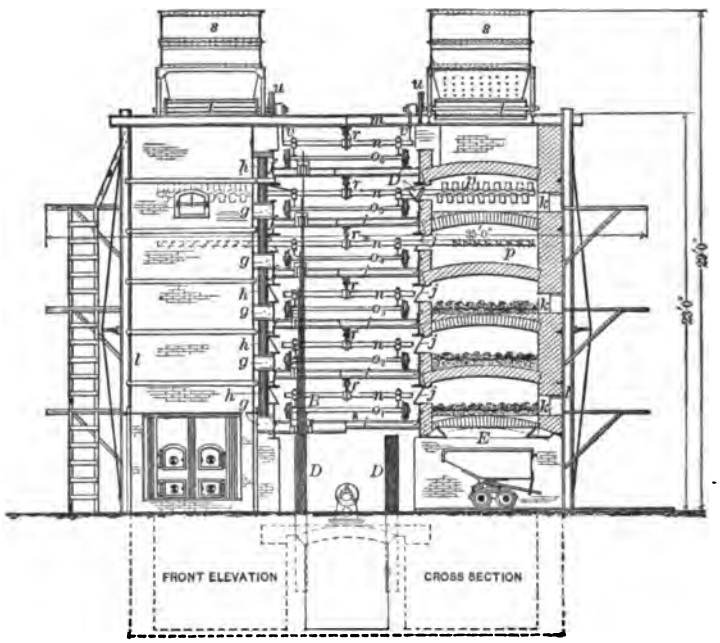


FIG. 48. KELLER FURNACE.

furnace is, laid endwise; consequently each hearth is 8 in. thick at the center. It is 20 in. from the hearth to spring of the arch and the latter rises 8 in. The surface of the hearths is leveled off with sand, ashes or ore. The top of the furnace is used as a drying hearth and is provided with a raking mechanism like those for the hearths below.

The rakes in this furnace are moved forward and back, as in the Spence furnace, but only the rakes proper are in the roasting chambers, the supporting carriage being outside, which of course necessitates a horizontal slot in the side wall. This is provided by building the walls of the pair of

furnaces, which face each other, in sections against 24 in. I beams, one of the latter to each hearth. The skewback of each arch rests on the lower flange of the I beam and the wall is built up to the upper flange. The 24 in. I beams are supported at the ends of the furnace on 12 in. I beams built into the end walls; they receive additional support from the cast iron columns *h*. There is thus formed a continuous slot, corresponding to each hearth, in each inner wall of the furnace.

The rake carriages, one for each pair of hearths of the same level, travel on rails supported by a steel framework between the two furnaces. A stirring arm projects into the hearth rooms on each side. The arms are made of 4 in. wrought iron pipe and may be cooled internally either by air

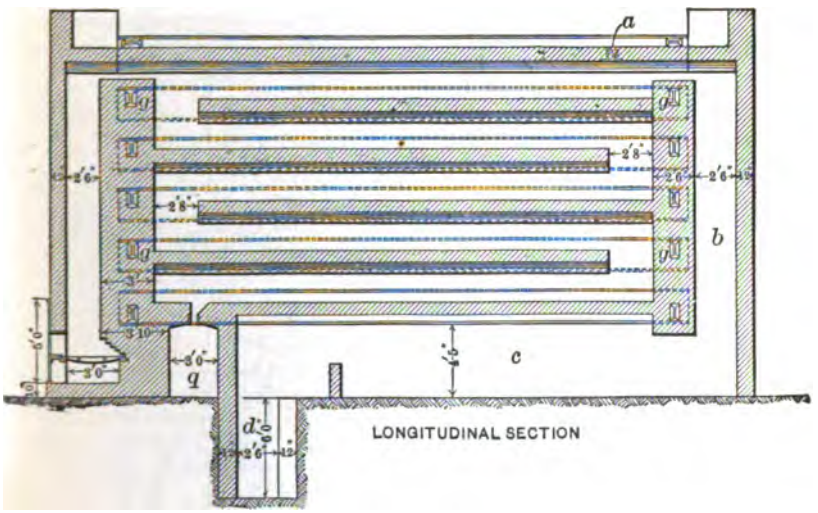


FIG. 49. KELLER FURNACE.

or by water. Two sets of rakes, 180° apart, are fixed to each arm; the purpose of this is explained further on. The carriages are moved by wire ropes, which pass over the necessary sheaves and thence to a large driving wheel, on the same shaft with a pinion that receives reciprocal motion from a rack actuated by a hydraulic piston.

The rake arms with their plows are turned by an ingenious mechanism so that the plows dig into the ore only when moving in the direction that the ore has to go. At each end of the furnace a four-point star wheel on the carriage strikes a tumbler, which turns the arms 90°. Consequently, when the plows have furrowed through the ore in moving forward, they go

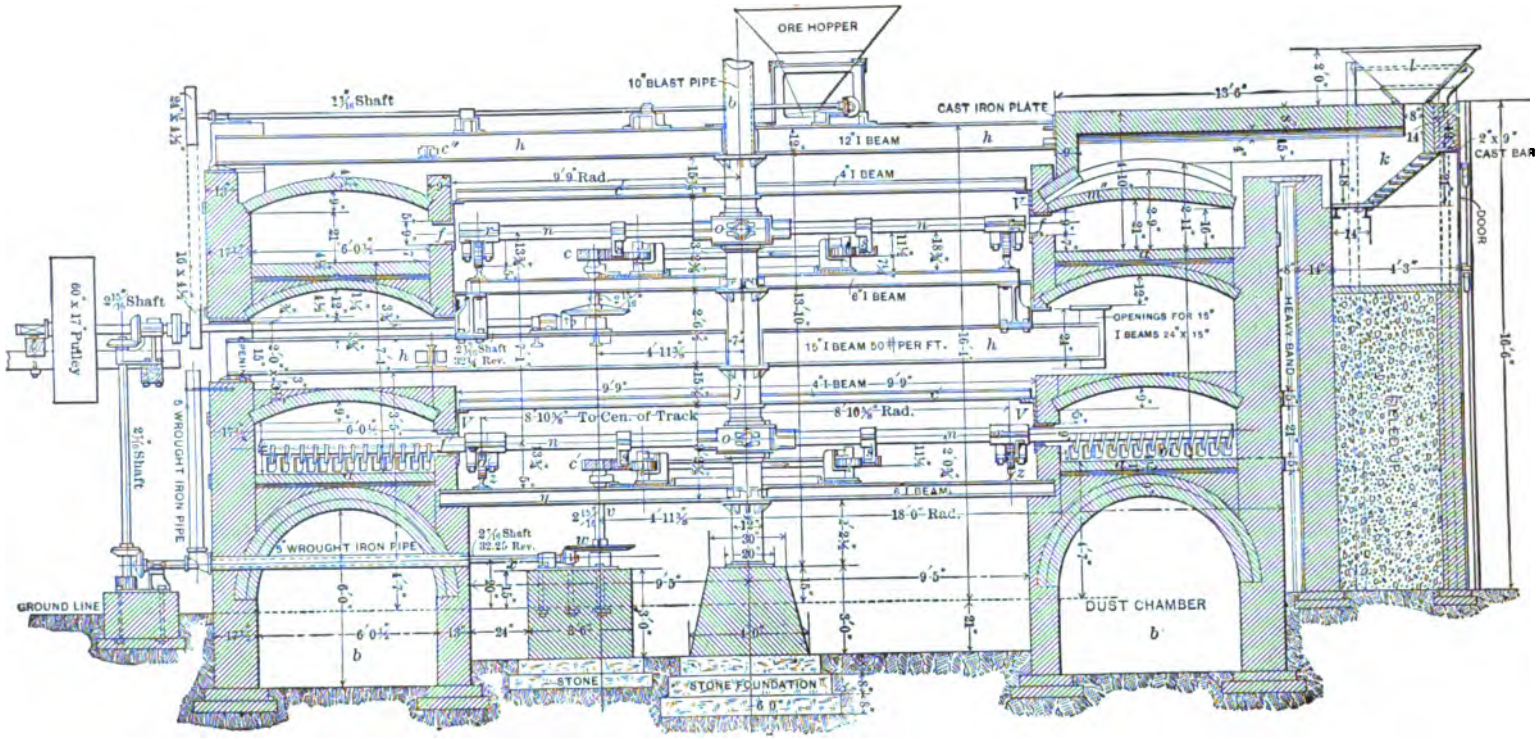


FIG. 50. PEARCE TURRET FURNACE.

Section through A B C on Fig. 51.

backward in a horizontal position over the ore. The plows on each side of the rake arm are used alternately and they are set so that each time they turn down the furrows of the previous movement.

The slots in the side walls of the furnace are closed by steel bands, wound and unwound on and from pulleys at each end of the furnace by the movement of the rake carriage; the pulleys are governed by springs so as to keep the bands taut and the winding is assisted by counterweights. Otherwise, the slots are closed by tripping doors as in the Wethey furnace. In the opposite walls of each hearth chamber there are doors, which give access to it when repairs have to be made.

Each one of the pair of furnaces has a fireplace with grate 2·5×4 ft. The flames rise through a vertical flue in the end wall and pass over the uppermost hearth; thence through a vertical flue in the opposite end wall into a flue beneath the lowest hearth and from there to the chimney. This is only to ignite more quickly the cold ore; after it is once well afire it (if pyrites) burns out sufficiently for the purposes of copper smelting. To effect a dead roast it would be of course entirely feasible to cause the flames first to traverse the lowest hearth. This furnace has not yet been used for blende roasting, but has given excellent results with copper ore.

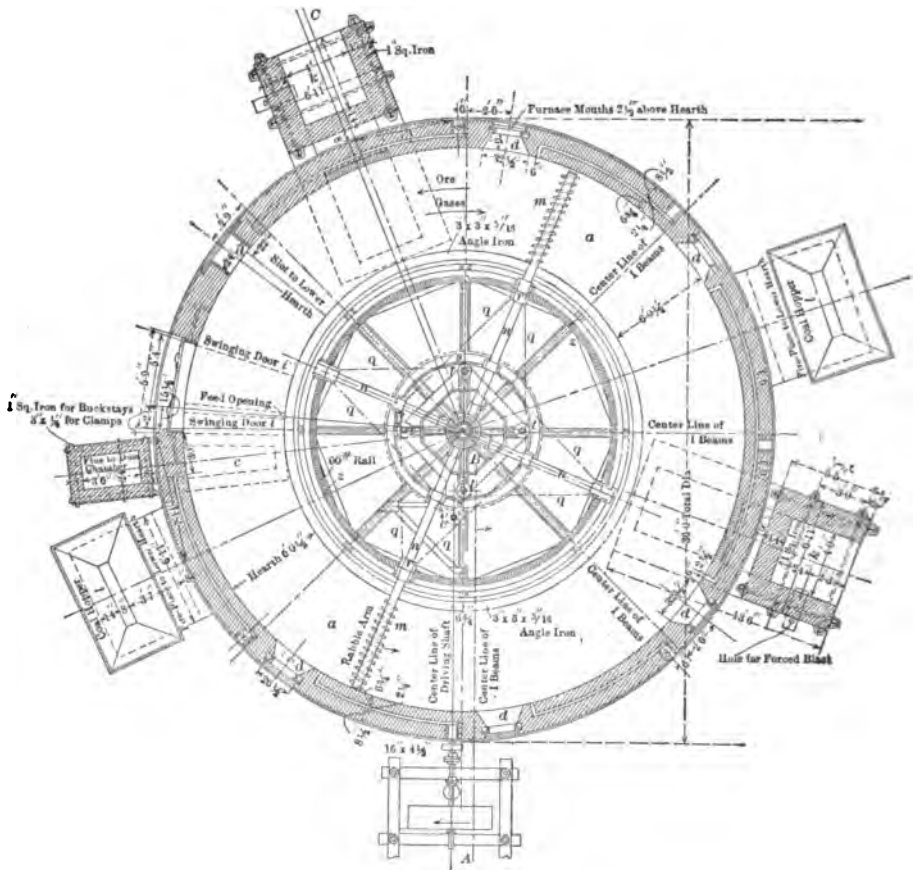
The Keller furnace (double) has a total hearth area of 1340 sq. ft. It costs $10,000@$12,000, there being required in its construction 130,000 lb. of iron, 160,000 red brick and 200 fire brick. It requires 2 to 3 h. p. One man per shift can attend to three furnaces. The repairs and renewals, of which the chief item is plow blades, are very light, amounting to only about $500 per annum.

Pearce Furnace.[1]—The Pearce "turret" furnace is an annular reverberatory, like the Brown "horseshoe," but its diameter is less and its construction and method of raking the ore charge are radically different. A segment of the hearth circle is left uncovered; at that point the roasted ore is discharged.

In the center of the circle, stands an iron column, which acts as a vertical axle for the revolving framework, which carries four pipe arms, radiating horizontally from the column and projecting through the inner wall of the hearth. On the end of the pipe arms are blades for the rabbling of the ore. To provide for the unobstructed sweep of the pipe arms, the inner wall of the hearth must have a continuous slot. This is accomplished by hanging the supports for the inner circumference of the furnace arch from beams, which rest, one end on the central column and the other on the outer wall of the furnace. The slot through which the pipe arms project is

[1] This furnace is built by the Stearns-Roger Mfg. Co., of Denver, Colo.

covered with a sheet steel shield, fastened to and rotating with the pipe arms. The brick work is braced from the inner wall to the central column by radial I beam struts and the outer wall is encircled by steel bands. The frame carrying the arms is rotated by cog gearing. At one end of the


Furnace Mouths 2½" above Hearth
Ore
Gases
Slot to Lower Hearth
Swinging Door
Feed Opening
Swinging Door
Center Line of I Beams
Center Line of I Beams
Hearth
Rabble Arm
Center Line of Driving Shaft
Center Line of I Beams
Coal Hopper
Coal Hopper
Coal Bin
Coal Bin
Coal Bin

ence, the entering flame being controlled and directed by curtain arches. The fireplaces are built to burn coal, wood, oil, or gas. Under the hearth, the space may be utilized as a dust flue.

It is obvious that the construction of the Pearce furnace permits multiplication in the number of hearths, but so far nothing beyond a two-hearth furnace has been built, and it would probably be not advantageous to increase the number further, although the two-hearth furnace is undoubtedly more economical in fuel than the single hearth.

The Pearce furnace is built in three sizes: (1) 36 ft. in diameter outside, hearth 6 ft. wide and 505 sq. ft. effective roasting area; (2) 38 ft. diameter, 7 ft. width of hearth and 609 sq. ft. area; (3) 40 ft. diameter, 8 ft. width of hearth and 788 sq. ft. area. The double hearth furnaces have twice the hearth area of the single furnaces. The fireplaces project 6 ft., so the 36 ft. furnace requires floor space of 36×42 ft. The single hearth furnace is 11 ft. high; the double hearth 16 ft. 6 in. The single hearth, 36 ft. furnace contains 40,000 lb. of iron, 65,000 red brick and 3500 fire brick. Before the rise in the prices of material in 1899 its cost erected at Denver, Colo., was $5500@$6500, including royalties. The 38 ft. furnace requires 50,000 lb. of iron, 90,000 red brick and 4000 fire brick. The high prices of 1899 increased the cost considerably, probably as much as 25%. A two-hearth 38 ft. furnace if erected in Kansas or Missouri in 1899 would doubtless have cost $12,000; a two-hearth 40 ft. furnace, $15,000. These estimates do not include dust chamber, chimney, engine, or houses. A plant of four 40 ft. furnaces would require a building about 200×60 ft., the furnaces being set in one row.

In the latest designs of the Pearce furnace the dust chamber under the hearth has been abandoned, the furnace being built solidly to the hearth, thus reducing the cost somewhat. Instead of four rake arms only two now extend into the furnace, the other two being held in reserve. The fireplaces are provided with flat grates or step grates, according to the character of the coal used. There are usually three fireplaces.

The stirring mechanism of the Pearce furnace is very simple. The arms extending from the central axis are made of two pieces of 5 in. pipe, the outer extension bearing the rakes. The latter are made in two forms, namely, rectangular sections of steel plate, ¼ or ⅜ in. thick, set vertically on an angle with the line of the arms; or pieces of bar iron set vertically and clamped to the arms. When necessary to replace worn-out rakes the section of the arm which carries them is exchanged for another previously prepared, the operation requiring only a few minutes. The rakes are graduated in length and direction from the inner to the outer circle, so

that the ore on the outer circle, which has to travel a greater distance, may remain in line with that near the inner circle. Great care is required in setting the rakes, since any wrong adjustment would tend to throw the ore outward or inward and an imperfect roasting would be the result. The slot through which the arms sweep is closed by a band of sheet steel, 12 in. wide, which is carried by the arms themselves and pressed against the slot by weighted bell cranks attached to a circular angle iron also carried by the rake arms.

The rakes move at an average speed of 75 ft. per minute, measured on the middle line of the hearth, wherefore in the 36 ft. furnace they make one revolution in 53 seconds and cause the ore to be stirred once in 26·5 seconds. The feed of ore from the supply hopper is controlled by a mechanism actuated from the main driving shaft of the furnace. The rake after passing over the discharge opening, in the uncovered segment of the hearth, arrives at the ore newly fed, spreads it out, and passes into the covered hearth, which is closed by an iron, swinging door. By an ingenious arrangement the air passing through the ports in the rake arms into the hearth chamber is cut off until it reaches the point where the ore is beginning to roast, since otherwise the comparatively cold air would retard the heating of the ore.

The Pearce furnace requires from 2 to 5 h. p., according to size, and the labor of one man per 12 hours, who is able to attend to a double hearth 40 ft. furnace as well as a single hearth 36 ft. furnace; indeed, one man per shift may attend to two furnaces, the only work to be done being to oil the machinery, stoke the fires and supervise the roasting. Repairs are low, comprising chiefly the cost of changing rakes once in four to six weeks and renewing rake arms once a year. The quantity of oil required for lubrication of the machinery will hardly ever exceed 0·025 gallon per ton of ore and may be a good deal less.

At the works of the Empire Zinc Co., at Joplin, Mo., a two-hearth 38 ft. Pearce furnace, 1218 sq. ft. hearth area, roasted 10 to 11 tons (20,000 to 22,000 lb.) of blende assaying about 30% S to less than 1% S per 24 hours. The furnace was attended by one man per shift. The construction of the double hearth furnace is substantially the same as that of the single hearth. At Joplin there were three fireplaces for each hearth and step grates were employed.

Ropp Furnace.[1]—The Ropp "straight line" furnace is one of the simplest of the mechanical roasters. It is in brief a long reverberatory furnace over the hearth of which a series of plows or rakes is drawn by a continuous

[1] This furnace is built by Harron, Rickard & McCone, of San Francisco, Cal.

cable, moving the ore steadily from the feed to the discharge end. Fire boxes are disposed at intervals along the sides of the furnace in order to maintain the requisite temperature throughout the great length of the hearth. Ropp's inventions pertain especially to the structural details of the furnace; the fundamental principle being common to several mechanical roasting furnaces. The Ropp furnace has the merits of simplicity in mechanism, moderate first cost and small cost of operation (including repairs).

The furnace is built up from its foundations much the same as an ordinary long reverberatory furnace, but under the hearth there is made a longitudinal tunnel which communicates with the chamber above the hearth by a slot 1·5 in. wide. A continuous tramway passes to one side of the furnace and through the tunnel under the hearth. On this tramway run a series of four-wheel trucks, disposed equidistantly apart on the endless

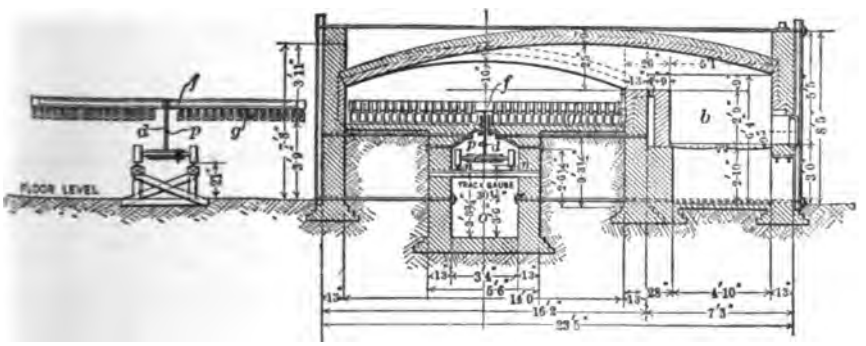
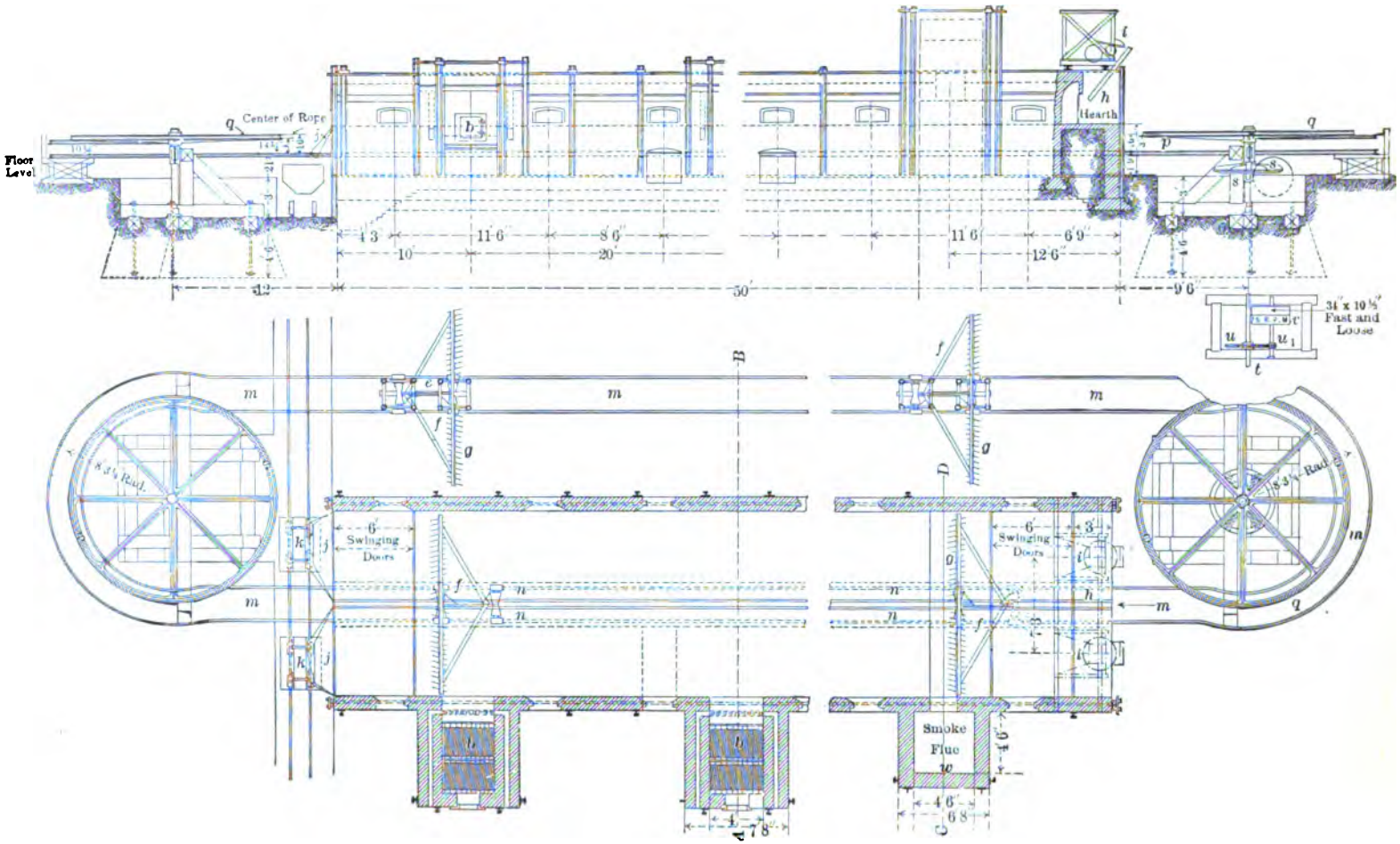


FIG. 52. ROPP FURNACE.

Vertical section through fire box on line A B of Fig. 54.

cable by which they are drawn, each truck having a vertical arm, which protrudes through the slot in the hearth of the furnace when the car is in the tunnel and carries a horizontal rake over the hearth, the rake being of the same width as the latter; the plows or blades of the rake are set at an angle of 45°. The rake enters the furnace at the feed end, moves the ore forward slowly and discharges it at the other end into a car or wheelbarrow. The ends of the furnace are closed by swinging iron doors, which are automatically pushed up by the rake when passing through. On leaving the furnace the truck and rake pass around on the outside section of the tramway and reënter at the feed end. In the tunnel there is sufficient room under the tramway for a man to enter to make any repairs that may be necessary. The traction rope, which is of steel wire and ⅝ in. diameter passes around large horizontal sheaves at each end of the furnace, power



FIGS. 53 AND 54. ROPP FURNACE, LONGITUDINAL ELEVATION AND PLAN.

being communicated to one of them through a bevel gearing and spur gearing from a horizontal shaft. The latter makes 75 r. p. m., which by the two sets of gearing is reduced to 1·63 r. p. m. for the rope sheaves. The distance traveled by the rope, truck and rake in the open air is about 1·5 times as long as in the furnace, so the various parts exposed to high heat have time to cool. The rope does not become hotter than the hand can bear and lasts from 18 to 20 months. The ore is fed regularly into the furnace by means of a pair of Hendy Challenge feeders, but, of course, any other mechanical feeding device may be used if desired. The heat is supplied from fireplaces along the side of the furnace, which have each a perforated fire bridge to insure complete combustion with air preheated in the side walls. Along the sides of the furnace there are openings through which the admission of air may be regulated and the process of roasting observed. When the furnace is heated by natural gas, as at Iola and La Harpe, Kan.,

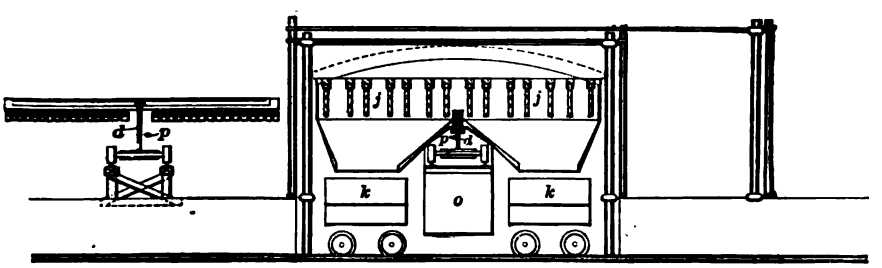


FIG. 55. ROPP FURNACE.
Elevation of discharge end.

the burners are protruded through these openings and there are no fireplaces. The products of combustion pass off to the chimney through a flue near the feed end of the furnace.

The method of buckstaying and tieing the furnace will be seen from the accompanying engravings. Expansion and contraction, which is a very important consideration in such a long furnace, is provided for by dividing the furnace into sections, interposed between which and forming a continuation of the main side walls are iron channel bars made to slide one within the other, but overlapping so as to give lateral stability to the walls; this permits each section of the furnace to expand or contract independently, and the proper form of the structure is preserved. Similar provision is made for the hearth and roof.

The Ropp furnace is built in various sizes, 150×14 ft. hearth dimensions being the most common. Other sizes are 50×14 ft., 100×14 ft., 105×11

ft., 132×14 ft. and 159×14 ft. The cost of the iron work of the 100×14 ft. and 150×14 ft. furnaces, f. o. b. San Francisco, and the quantity of brick and other material required were reported by the manufacturer, early in 1899, as follows:

Item.	100 ft. × 14 ft.		150 ft. × 14 ft.	
	Quantity.	Cost.	Quantity.	Cost.
Iron work, exclusive of buckstays *a*.	71,600 lb.	$5,800	83,500 lb.	$6,260
Buckstays, 6" I beams, 12¼ lb. per ft. . .	8,400 "	200	10,500 "	240
Red brick.	126,000	. . . . .	175,000	. . . . .
Fire brick, standard.	3,700	. . . . .	4,500	. . . . .
" " arch.	3,300	. . . . .	6,500	. . . . .
Lime.	90 bbls.	. . . . .	145 bbls.	. . . . .
Fire clay.	25 "	. . . . .	45 "	. . . . .

a Includes traction rope, and two iron frame, automatic ore feeders.

In June, 1899, the cost of the iron work had increased 15% upon the figures quoted above. Inventor's royalty is included in the cost of the iron work. The cost of labor for erecting a 150×14 ft. furnace may be reckoned

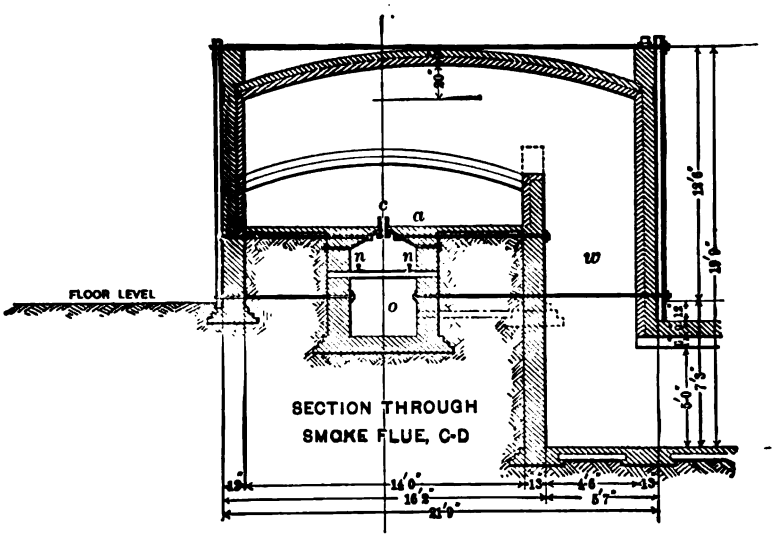


FIG. 56. ROPP FURNACE.

Vertical section on line C D of Fig. 5 .

about $850. The cost of a 150×14 ft. furnace complete will probably be $8000@$9000, according to the prices for material.

The 100×14 ft. furnace requires a chimney 4×4 ft. in area; while the 150×14 ft. furnace should have a chimney 4 ft. 6 in.×4 ft. 6 in. The

manufacturer of the furnace recommends that the flue area be made about 25% larger than the chimney area. The 100×14 ft. furnace for coal firing has three fireplaces and four rake cars; the 150×14 ft. furnace has four fireplaces and six rake cars.

The right to the use of the Ropp furnace for blende roasting in Kansas and Missouri is controlled by the Lanyon Zinc Co., which has a large number of them in use at its works at Iola and La Harpe.[1] The furnaces there are of standard construction, except that there are no fire boxes, natural gas being employed as fuel; only one rake car is used, which after being run through the furnace is stopped on the return track and allowed to cool; and the slot irons were faced with tiling in accordance with the wishes of the managers of the works, though the manufacturer considers this an unnecessary precaution even in view of the high heat attained in blende roasting.

The cost of roasting with the Ropp furnace is low inasmuch as one man per 12 hours can easily do all the work about it that is required. The cost of delivering the ore into the feed hoppers and taking it away from the discharge end will depend of course upon the manner in which that work is systematized. The bill for repairs is comparatively light since the cable and the rakes are exposed to high heat only part of the time. The former may be expected to last 18 to 20 months and the latter 10 to 12 months. The 150×14 ft. furnace requires 6 h. p. There are no available data as to the consumption of fuel required in blende roasting. The furnaces of the Lanyon Zinc Co., which are 150×14 ft., roast per 24 hours from 18 to 21 tons of raw blende assaying about 30% S, yielding 15 to 17·5 tons of product assaying 0·75% to 0·9% S.

Data as to Ropp furnaces of various dimensions are summarized in the following table:

Dimensions of hearth	105 ft. × 11 ft.	100 ft. × 14 ft.	150 ft. × 14 ft.
Hearth area, deducting slot	1,120 sq. ft.	1,367 sq. ft.	2,050 sq. ft.
Number of fireplaces	3	3	4
Length of single grate *a*	4 ft. 6 in.	5 ft.	5 ft.
Width of single grate *a*	4 ft.	4 ft.	4 ft.
Grate area, square feet *a*	18	20	20
Ratio hearth to grate area *b*	64:1	70:1	105:1
Number of rakes	4	4	6
Number of teeth per rake	26	32	32
Time of circuit, minutes	3·5	3·5	5·3
Ore stirred every (seconds)	52	53	53
Horse power required	4	5	6
Number of men per 12 hours	½ or 1	½ or 1	½ or 1

a These dimensions would be increased for a blende roasting furnace or the number of fireplaces would be increased. *b* This ratio would be diminished.

[1] These have lately been altered into Cappeau furnaces (vide the foot-note to p. 96; also the foot-note to p. 85).

Ross & Welter Furnace.—The Ross & Welter furnace,[1] which was once employed at Oberhausen, Germany, consisted of three superimposed hearths, of which the upper two were circular and the lowest was rectangular like that of the ordinary reverberatory furnace. The ore on the circular hearths was stirred by rabbles secured to a sleeve, through which passed a vertical hollow cast iron shaft. The latter was driven by a suitable bevel gearing, making one revolution per minute. Air was made to pass through the hollow shaft in order to cool it. On the lowest hearth the ore was moved forward by hand. With this furnace one man per shift was able to roast

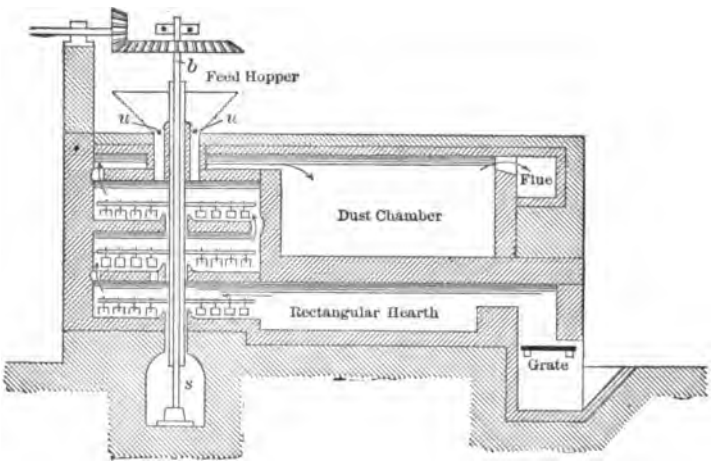


FIG. 57. ROSS & WELTER FURNACE.

3000 kg. of blende per 24 hours with a consumption of 750 kg. of coal. The hearth area was 22 sq. m. (236·5 sq. ft.) and the grate area 1·4 sq. m.[2] The rate of roasting was therefore approximately 28 lb. per sq. ft. of hearth per 24 hours.

This furnace was obviously an adaptation of the McDougal, which was invented some 20 years ago but did not come into extensive use until recently, when the mechanical improvements of Herreshoff, O'Brien, Klepetko & Evans, Trent and others developed forms which have proved to be highly efficient in the roasting of pyrites and copper ores.

Vieille Montagne Furnace.—The Société Anonyme de la Vieille Montagne patented a mechanical roasting furnace very much like that of Ross & Welter, except that it had three circular hearths instead of two. Its design is illustrated in the accompanying engraving. The products of com-

[1] Mr. Ross was director of the works of the Société Anonyme de la Vieille Montagne at Oberhausen, and Mr. Welter was an engineer there. The blende roasting plant at Oberhausen is an important one.

[2] P. Mahler, Annales des Mines, VII, 512.

bustion pass over the rectangular hearth and back and forth over the circular hearths in the direction shown by the arrows, escaping from the uppermost hearth into a dust chamber and thence to the chimney flue. The ore is fed on the topmost hearth from the hopper above it by means of the small fluted rollers which are shown in the illustration. The vertical shaft carrying the rabbles is surrounded by a cast iron cylinder, which is so attached to the shaft that it rotates with it. Air circulates between the shaft and its jacket in order to keep the shaft cool. The rabble arms are attached to a sleeve made in two halves, which embraces the cast iron jacket of the shaft. On each hearth there are rabble arms, one of which is provided with tines while the other has inclined blades; the former stir the ore and the latter push it, according to their inclination, either toward the middle or toward the circumference of the hearth, where it falls through holes to the hearth next below. This furnace has also been used at Oberhausen.[1]

Wethey Furnace.—The Wethey furnace is a mechanically raked furnace with a long rectangular hearth, similar in principle and general design to the Ropp and Brown straight line furnaces. The distinctive feature of the Wethey furnace is in the method of supporting the roof arch, the skewbacks of which are I-beams hung from horizontal transverse I-beams and held in place by short struts riveted to I-beam buckstaves. This method makes it possible to arrange a slot the whole length of the furnace in each side wall, and the trucks which carry the stirring mechanism can be made thereby to run on rails outside of the roasting hearth, with the advantage that they are not subjected to the intense heat of the inside. The slots in the side walls are closed by small overlapping pieces of sheet steel, each pivoted at the upper corner, so that the axle of the stirring carriage can lift them during its passage. As soon as the carriage has passed, the pivoted sheets drop again to their natural position. The stirring carriage is drawn by chains which pass over large sheaves at each end of the furnace, so that the carriage after passing out of the roasting chamber passes back on the rails above the furnace. The general construction of the Wethey furnace is illustrated in the cross section shown in the accompanying engraving. The furnace is heated from fire boxes disposed on each side, varying in number and position according to the length of the furnace and the character of the ore.

A Wethey furnace for roasting zinc blende was erected by the Empire Zinc Co. at its works at Joplin, Mo. It had a hearth 12 ft. wide and 164 ft. long, giving a hearth area of 1968 sq. ft. The fire boxes on the sides were fitted with step grates and were operated with an under grate blast. The products of combustion were exhausted from the furnace by a brick chimney

[1] German patent No. 24,155.

125 ft. high. It was said that with a crew of two men per shift this furnace would roast 20 tons of ore per 24 hours, which is 20·3 lb. per sq. ft. of hearth per 24 hours. The Wethey furnace at Joplin was pronounced a failure, which was ascribed by its designers to alterations from their plans that were made by the company for whom the furnace was built, and not because of the inadaptability of the furnace to blende roasting. I am disposed to consider that this claim is correct, for I can see no reason why under proper conditions the Wethey furnace should not give as good results in roasting zinc blende as some other mechanical furnaces which have been

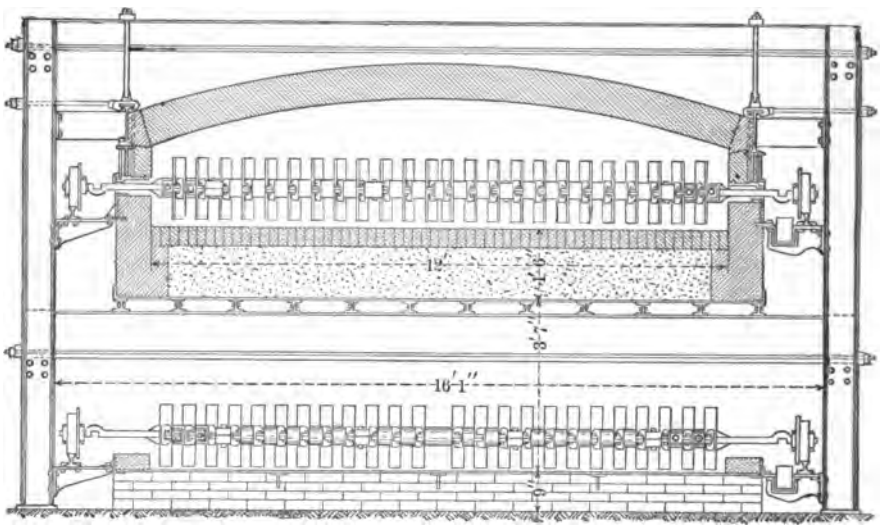


FIG. 58. WETHEY FURNACE.
Tranhverse section through hearth. Scale, ¼ in. = 1 ft.

successful. The imperfect manner in which the side slots are closed, however, by the tripping doors appears to me a serious defect in the Wethey furnace, inasmuch as the leakage of air through them must prevent any close control of the air supplied for oxidation. The doors are apt to give a good deal of trouble moreover by becoming warped and failing to drop after passage of the rake carriage.[1]

Zellweger Furnace.—This is a long hearth, mechanical reverberatory furnace, which has been adopted at the works of the Cherokee-Lanyon Spelter Co. near Iola, Kan. As exemplified at those works the furnace has

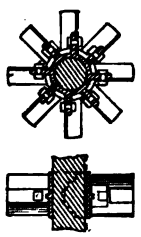
[1] The Wethey furnace was formerly built by the E. P. Allis Co., of Milwaukee, Wis.; now by the Allis-Chalmers Co.

an effective hearth area of 15×135 ft. and is heated by natural gas, which is introduced at intervals through openings in the side walls. The particular feature of the Zellweger furnace is the raking mechanism, which consists of a heavy shaft, supported at its ends by two fixed wheels and carrying a number of collars on which are mounted radially projecting blades of a smaller reach than the radius of the supporting wheels. There is formed thus a cylinder of blades of about 2 ft. diameter, which extends crossways over the hearth and nearly touches it. The wheels have a diameter of 6 ft. and roll on tracks in the bottom of wheel pits on each side of the hearth. When the stirrer rolls over the hearth the point of each blade, moving in a cycloidal curve, in turn approaches to and then recedes from the hearth. The collars are mounted loosely on the shaft, but are provided with a locking device, so that they can revolve around the shaft when the stirrer rolls in the direction toward the feed end of the furnace, but are locked to the shaft and cannot revolve around it when the stirrer rolls in the direction toward the discharge end. The stirrer blades are made of steel plates formed concave in front and convex in rear, showing a V or U-shaped cross section. During the forward trips of the stirrer over the hearth in the direction toward the discharge end, each blade, being fixed on the shaft by the locking device, digs into the bed of ore and with its end scrapes along the hearth. In doing so the cavities in the front of the blades become filled with ore from the bottom and as the blades rise they drag this ore forward and upward and spread it over the top of the charge. This action produces a thorough stirring and mixing of the entire charge. During the return trip of the stirrer over the hearth, toward the feed end, the blades also dig into the bed of ore, but since the collars are then free to revolve around the shaft, each set of them is caused to roll over the hearth like a wheel on its track, the blades in this case only displacing the ore laterally like a plow or rake. It is obvious that the stirrer on its forward trip carries much more ore forward than it moves backward during its return trip, and that this difference measures the rate at which the entire charge advances over the hearth toward the discharge end.

The tracks in the wheel pits consist each of a line of rails or beams and a line of racks all bolted together. They are cooled by an air channel beneath them, the air so heated being drawn into the roasting chamber. The teeth of the racks correspond to and engage with pockets on the circumference of the stirrer wheels, thereby securing parallel motion of the stirrer. The stirrer is set in motion by means of an endless chain of which one half is supported by idlers in the bottom of one of the wheel pits and the other half by idlers on top of the furnace. The chain is attached to the stirrer

by means of a collar which contains a ball-bearing and is mounted on the shaft near one of the wheels. The chain runs over sprocket wheels, located outside of each end of the furnace. One of the sprocket wheels is driven by an automatic reversing engine which pulls the stirrer forward and backward through the furnace, starting and stopping itself after each single trip. The stirrer passes through the furnace in 1·5 minutes and then stops for 1·5 to 3 minutes as desired. At the Cherokee-Lanyon works the pause at each end of the furnace is two minutes. The furnace chamber is closed at each end by fixed, hanging walls and swinging sheet iron doors.

At the feed end of the furnace the ore is drawn automatically from a hopper to a movable platform, which is an extension of the hearth. This platform is ordinarily dropped below the level of the hearth, so the ore is out of reach of the stirrer when the latter rolls over it on the backward trip, but before the stirrer starts forward the platform is raised to the hearth level by means of hydraulic pistons, so the stirrer engages the fresh quantity of ore which has been drawn from the hopper. At the discharge end of

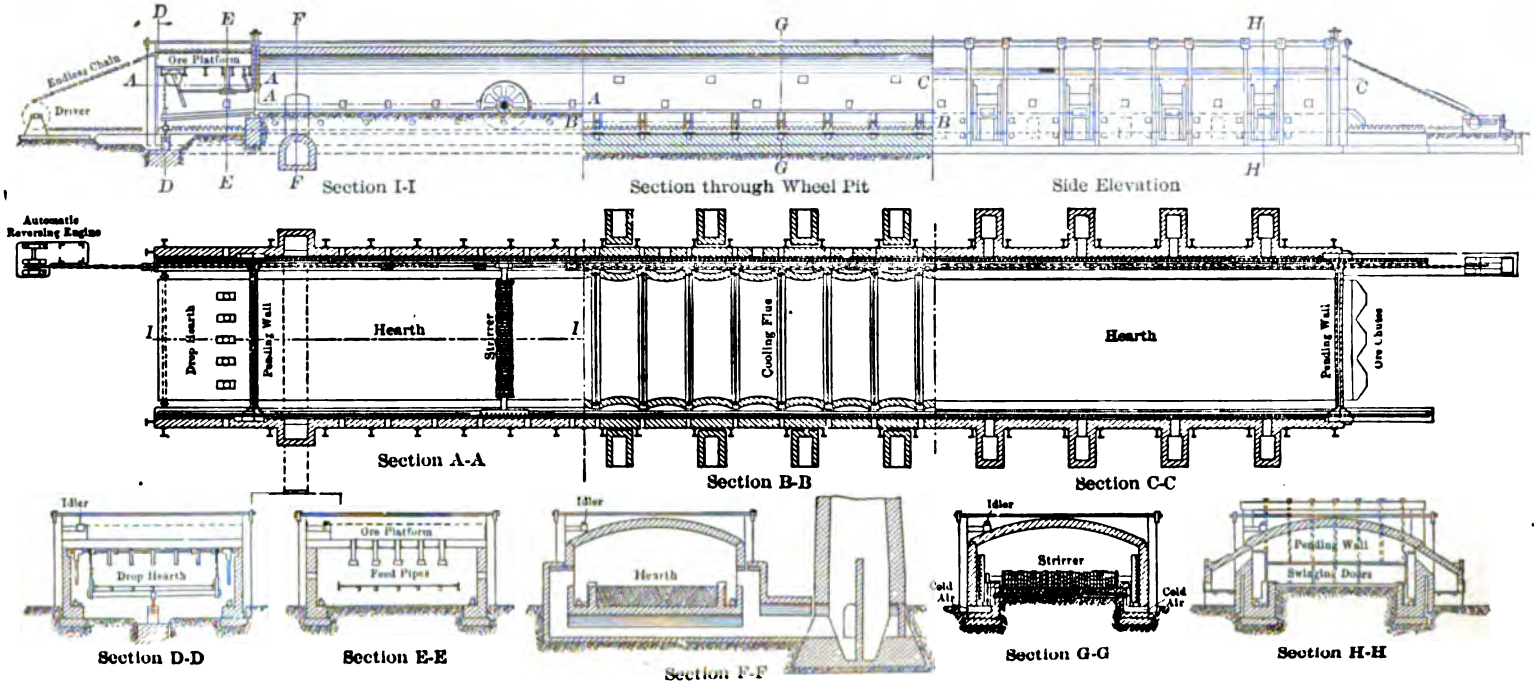


FIGS. 66 AND 67.
ZELLWEGER FURNACE.
Sections through stirrer blades and shaft.

the furnace the roasted ore is simply pushed off the hearth into a chute leading to a car or wheelbarrow.

The connection of the roasting chamber with the chimney will be seen in the accompanying engravings. It will be observed from the transverse sections that the height from the hearth to the roof arch is necessarily greater than in other reverberatory furnaces, in order to give clearance for the large wheels of the revolving rake. If the furnace were to be used with coal firing the grates would be disposed along the sides, as in other roasters of this type. The furnace shown in the accompanying engravings is designed for heating by natural gas.

The Zellweger furnace at the Cherokee-Lanyon works roasts 18 tons (36,000 lb.) of ore assaying 28 to 30% S down to 0·8 to 1% per 24 hours. If the desulphurization be carried to 0·5% the capacity of the furnace is reduced to 15 tons, while about 21 tons of raw ore can be put through per 24 hours yielding a product which assays 1·5%.



FIGS. 59 TO 65. ZELLWEGER FURNACE.

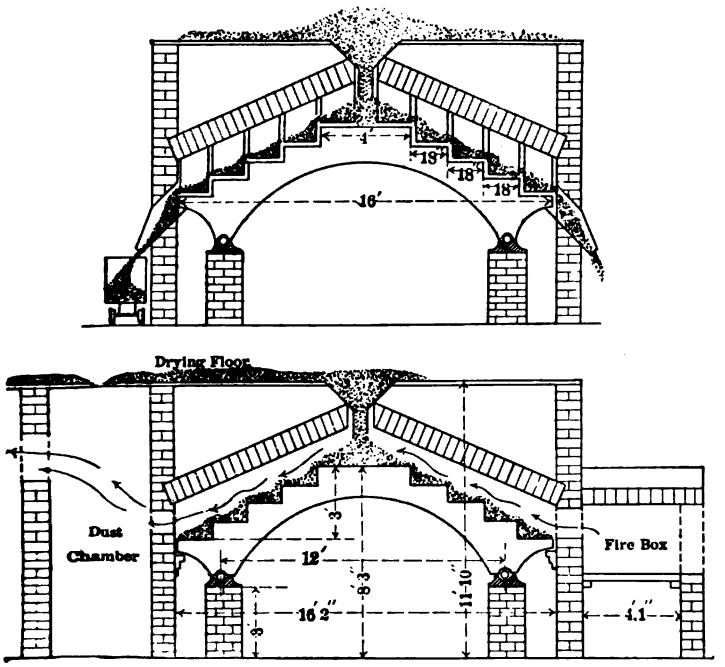
FURNACES WITH MOVABLE HEARTHS.—Mechanically raked furnaces with movable hearths are comparatively unimportant as a class. Several varieties have been employed, however, in blende roasting. When the hearth is movable the rakes must be stationary. The hearth in this type is usually made circular, that being the form to which it is easiest to impart motion. It is mechanically feasible, however, to construct a long rectangular hearth with a reciprocating motion, as has been proposed by Argall in the form of furnace described in the following paragraph.

Argall Furnace.—This is a reverberatory furnace, of which the hearth has a reciprocating movement, back and forth, whereby the ore is caused to move forward by the action of fixed rabbles extending across the hearth from side to side. The rabbles are arranged so as to be lifted mechanically out of the ore, so as not to interfere with the advance of the latter when the hearth makes the return movement. The hearth consists of tiling laid on transverse I-beams, which rest on a pair of inverted T-rails. The latter move on a series of car wheels, which are fixed in position, but are free to turn. The hearth is made longer than the furnace proper by the extent of the movement conferred upon it, which may be 10 to 30 ft.; consequently, it projects alternately from one end or the other by that amount. The furnace may be of a length of 100 ft. or upward. The drawings accompanying the patent specification show a design in which the hearth has a movement of 25 ft., the rabbles being arranged 24 ft. apart. The hearth is moved at intervals of 10 to 30 minutes, according to the character of the ore.[1] The mechanical details of this furnace have been well worked out, but it has not yet received a practical trial.

Blake Furnace.—This furnace, which has so far been used only for the roasting of pyrites and blende, as a preliminary to the magnetic separation of those minerals, but is adaptable to a more complete desulphurizing roasting, is characterized by a circular revolving hearth and is a modification of the old Brunton furnace, used in Cornwall, from which it differs in various features. The hearth has the form of a series of annular terraces, rising from the outer edge to the center and is supported by cast iron balls running in a grooved circular track 12 ft. in diameter. The hearth, which is 16 ft. in diameter, is covered with fire brick; the terraces are 18 in. wide, except the uppermost, which is 4 ft. in diameter. The hearth is enclosed in a circular brick chamber with a dome shape roof, the general design being shown in the accompanying engravings, from which there will be no difficulty in understanding the operation of the furnace. The ore

[1] Philip Argall, United States patent No. 653,202, July 10, 1900. This furnace is not to be confused with Mr. Argall's well-known revolving cylinder furnace, which is described in a subsequent section of this chapter.

having been received on the flat roof, which serves as a drying floor, is delivered into the central hopper, whence it is drawn automatically to the uppermost terrace of the hearth. Plows are fixed in the dome shape roof, two over each terrace, in a line at right angles to the plane of Fig. 69 and as shown in Fig. 68. These plows being set at an angle of 45° turn the ore over in a succession of furrows as the hearth revolves, moving the ore constantly from the center outward, taking it first to the edge of the highest



FIGS. 68 AND 69. BLAKE REVOLVING FURNACE.

The hearth is circular in plan. Fig. 68 is a vertical section of the furnace through the discharge chutes. Fig. 69 is a vertical section 90° from the plane of Fig. 68.

Scale, approximately ⅛ in. = 1 ft.

terrace, from which it falls over to the next, within reach of the corresponding plow. By this, after one revolution of the table, the ore is turned over into a furrow or ridge so as to lie within reach of the next plow, by which the ore is again thrown off upon the next lower terrace, and so on, until it is discharged over the lowest terrace into a chute which delivers it into a car. The plows are so placed that the ore turned over by one remains undisturbed until it has made a complete revolution with the table

before it is turned by the next plow; or it may be turned over at each half revolution if desired. In Fig. 68 the ore is shown discharging at two sides, but it is preferable to discharge on one side only, the last plow on the opposite side being omitted; there may be only one plow to each terrace, but two are preferred, as in Fig. 68. The height of the plows above the hearth is adjusted through the dome, these details not being shown in the accompanying engravings. The speed of the hearth is designed to be 10 revolutions per hour, but can be varied to suit the ore, etc.

The fire box is placed at one side of the hearth chamber and the flames from it play freely across the hearth, escaping through a flue at the opposite side into a dust chamber. The air required for oxidation of the sulphur is preheated in brick checkerwork in separate chambers placed on opposite sides of the fire box, the checkerwork being heated by independent fireplaces. These stoves, as they may properly be called, are not shown in the accompanying engravings. The effect of hot air in ore roasting was discussed in Chapter III.

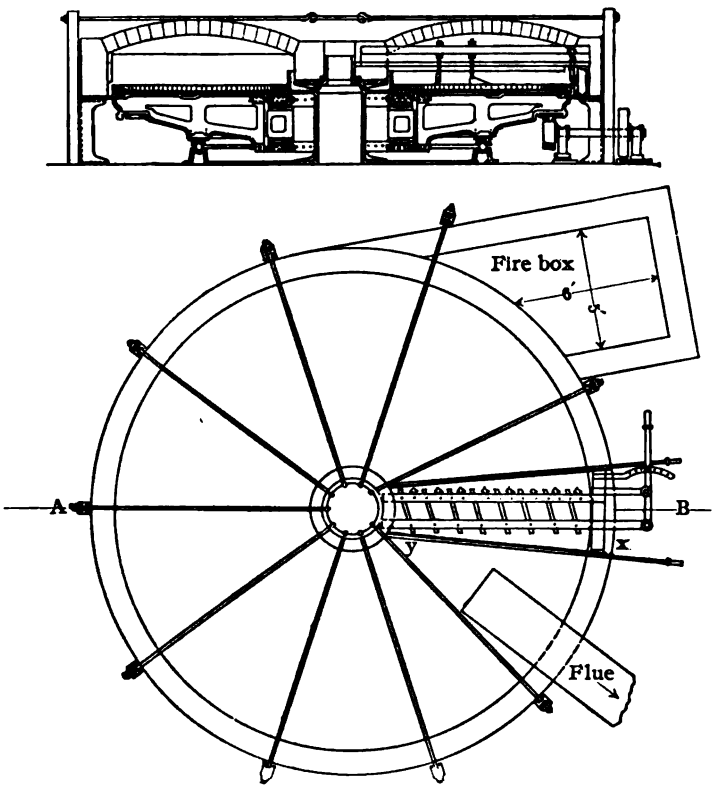
Godfrey Furnace.[1]—This furnace, which has an annular revolving hearth, was lately in use for roasting blende and mixed sulphide ore at the works of Fry, Everitt & Co., at Swansea, Wales, and those of the Smelting Corporation, Ltd., near Manchester, England. My description of it and the data concerning its results are taken from a paper by W. Blackmore, read before the Institution of Mining and Metallurgy, June 21, 1899, and a private communication from Mr. Blackmore.

As will be seen from the accompanying engraving, the Godfrey furnace consists essentially of an annular rotating hearth. The roof covers all the hearth with the exception of one sector, each portion of the rotating hearth coming in turn into this open sector. In this open space is placed a plow, the blades of which can be moved in any direction by a simple lever arrangement. The hearth of the furnace is built on cast iron cantilevers, of which there are 10, fixed firmly into central rings, and thus forming one rigid frame. On this frame are laid cast iron plates, and on those is the actual floor of the furnace, made of 2 in. fire clay slabs. The brickwork roof is thrown between the outer ring of masonry and the central column. The furnace is about 20 ft. in diameter, and as will be seen from the drawing, the frame of the hearth is supported on ball bearings, upon which it revolves, being driven by an ordinary tooth wheel gearing. The rate of travel is designed to be one revolution in from two to three minutes, but this may be increased or decreased as desired. Only 1 h. p. is necessary

[1] Patented by Joseph Godfrey and Henry J. Hayes, of Swansea, Wales; assignors to the Godfrey Calciner, Ltd., London, England; United States patent, No. 637,864.

to work the furnace. The opening in which the plow is fixed is 3 ft. 6 in. wide on the outside, tapering down to 2 ft. in the center. The ore is charged through a hole in the roof or a door in the outer wall, as may be deemed most convenient.

In a short time after the ore is charged it becomes uniformly red hot, for each part of the ore charge passes in its revolution through the hottest



FIGS. 70 AND 71. GODFREY FURNACE.
Fig. 70: Vertical section. Fig. 71: Plan. Scale, ⅛ in. = 1 ft.

portion of the furnace. During each revolution the ore is adequately turned by the plow, and thereby brought into contact with fresh air unmixed with carbon and sulphur gases. Thus, the furnace is one in which the heated ore is brought alternately first into the closed portion of the furnace in contact with the furnace gases, and secondly, into the open part away from the products of combustion altogether.

The plow being placed in the opening of the furnace naturally is subjected only to a comparatively low degree of heat. The plow is so arranged that it can be taken out and have new blades fixed in a very short time. When calcined sufficiently the ore is discharged automatically by moving the blades of the plow against the ore, and in the course of a few minutes the ore is all withdrawn from the furnace. This operation is effected without cooling down the furnace at all, and the next charge is introduced without delay on the still fully heated hearth.

The 20 ft. furnace has a hearth area of 280 sq. ft. and roasts 5·6 short tons or five long tons (11,200 lb.) of blende in 24 hours down to 2% S. The cost of roasting at Swansea is given as 5s. ($1·25) per long ton, including labor, fuel, power and repairs,[1] wages at Swansea for the class of labor required for such a furnace being 4s. ($1) per day and coal 7s. 6d. ($1·82) per long ton. In construction the Godfrey furnace requires 56,000 lb. of iron, 7000 red brick and 8000 fire brick. After the foundations are ready the furnace can be put up in about two weeks. It is highly important to have good concrete foundations to avoid any chance of sinking of the ball-track and to prevent jamming of the tooth-gearing. This precaution is of course very essential in any furnace with a movable hearth.

Revolving Cylinders.—These are practically a variety of mechanical furnaces with movable hearths, but are so different in their construction, if not in principle, that they are appropriately considered separately. Revolving cylinders of the reverberatory type have not yet been employed practically in blende roasting; the muffle type has been tried experimentally. Revolving cylinders are easy to manage and are efficient in roasting some kinds of ore, but they make more dust than other kinds of mechanical furnaces do, this being especially the case when the lining of cylinder is put in with projecting courses, the object of which is to lift the ore and shower it through the flame. The advantages claimed for cylindrical furnaces are that the cost of repairs and renewals is less than in other mechanical furnaces, there being no stirring mechanism exposed to intense heat, and the ore is stirred slowly and continuously, fresh particles being constantly turned up for oxidation. To promote the latter action the cylinder has been divided into segments by internal diaphragms, as in the Rothwell furnace, or instead of one large cylinder several small ones are united in a group and revolved together, as in the Argall furnace. These modifications probably increase the percentage of dust carried off. The dust forming tendency of revolving cylinders is

[1] It is doubtful what labor is included in this estimate. Some metallurgists include cost of delivering ore and coal to the furnace and removing ore and ashes in the cost of roasting; others include only the cost of labor in managing the furnace. Obviously the latter is the only method which will enable true comparisons.

not a very important objection to them, however, inasmuch as with a properly designed dust chamber it can be practically all recovered, and the only real objection is the additional quantity of that material which must be reworked. The rate at which ore will pass through a continuously discharging cylinder is, of course, governed chiefly by the angle of inclination of the latter and the speed of revolution. The capacity of an intermittently discharging cylinder, set horizontally, is dependent chiefly on its internal length and diameter, the speed of rotation, and the degree of desulphurization desired. The Argall, Brückner and White-Howell exemplify the various types.

Argall Tubular Furnace.[1]—The Argall tubular roasting furnace consists of four brick lined steel tubes, 30 ft. long, which are nested together inside of two steel tires, and are provided at the front and rear ends with heads which serve to receive the raw ore, and feed it into the tubes, and to discharge the roasted ore. The tires carrying the tubes rest on and are driven by steel-faced carrying rolls, which are driven by suitable gearing arranged so as to allow for any variation in the size of the steel tires or carrying rolls. The bearings of the carrying rolls are of the ball and socket type, lined with phosphor bronze, and are supported by cast iron sole plates resting on a heavy I-beam framework.

The roaster is set on a slight incline (the angle being varied according to time required for roasting) and is held in place by vertical guide rolls. The fire box consists of a fire brick lined steel shell, provided with a flue corresponding to the hole in the discharge end of the roaster. The shell is supported by a steel framed, wheeled truck so that it may be moved to one side when the roaster requires repairing.

This roaster is always in balance, the charge of ore in an ascending flue being counterpoised by the ore charge in the descending flue; consequently the power used is only that required to overcome the friction. The multitubular form of construction gives such great strength that two points of support are ample for a furnace 60 ft. in length, thus intermediate supports and their attendant friction are abolished. Wear is confined to the gears and friction drives, which are away from the heat.

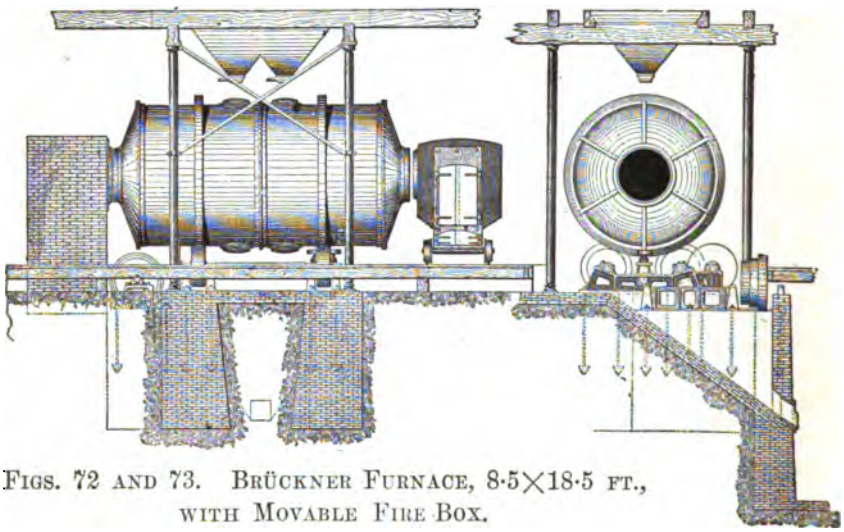
The Argall furnace requires 1 h. p. and occupies a floor space of 756 sq. ft. The iron work weighs 63,600 lb. and costs about $5000 f. o. b. Denver. In the construction of the furnace there are required 14 tons of fire clay flue lining, 5000 fire brick and 20,000 red brick, but if stone be used for the foundation piers 10,000 red brick will suffice.

Brückner Furnace.—This well known type of roaster consists of a horizontal cylinder of boiler iron lined with fire brick, the cylinder resting on

[1] Built by the F. M. Davis Iron Works Co., of Denver, Colo.

friction rollers and revolving between a fire box and a flue. The flame from the fire box passes directly through the cylinder, and thence, mixed with the gases from the ore, into a dust chamber. The cylinder is provided with manholes for receiving and discharging ore.

The cylinder is made to revolve slowly; in the case of small furnaces by applying power to a shaft carrying the friction rollers; in the case of large ones by a pinion which engages a spur-gear surrounding the cylinder. The fire box is made either movable or stationary. In the former case it is constructed in the form of a car running on a track, usually (but not invariably) placed at right angles to the axis of the cylinder, and having a



FIGS. 72 AND 73. BRÜCKNER FURNACE, 8·5×18·5 FT., WITH MOVABLE FIRE BOX.

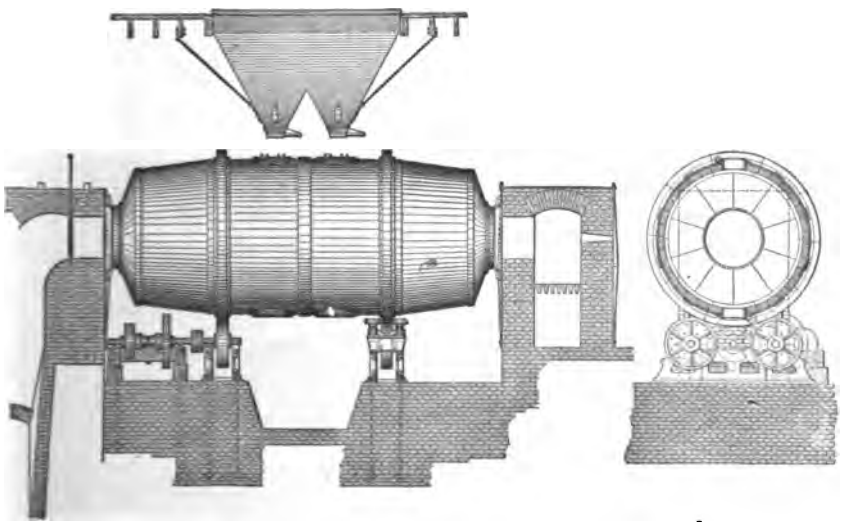
Fig. 72: Side elevation. Fig. 73: End elevation.

short flue on one side that comes exactly opposite the throat of the furnace. In this way the fire box can be run in front of a cylinder containing a fresh charge and fired until the sulphur is thoroughly ignited, and then run opposite another cylinder, leaving the first to complete the combustion of the sulphur with free access of air undiluted by carbon gases. Toward the end of the roasting when extraneous heat is required the fire box is moved opposite the throat of the furnace once more. In the case of ores which need extraneous heat throughout the operation a cylinder with a fixed fire box is used.

The cylinder is usually lined with red brick around the sides and fire brick at the ends, but sometimes the lining is made of fire brick throughout,

which are easier to put in on account of the greater regularity of their shape and are more durable. A red brick lining, when well put in, will last however about 18 months, and this is the principal cost for repairs and renewals during the first few years of the life of the furnace.

The furnaces shown in the accompanying engravings have four feeding and discharging openings, arranged in pairs half way around the circumference of the cylinder, so that the latter has to be turned only 180° in order to charge or discharge. The holes come directly under the chutes of a double-feed hopper. It is to be observed that the Brückner furnace dif-



FIGS. 74 AND 75. BRÜCKNER FURNACE, 7×18 FT., WITH STATIONARY FIRE BOX.

Fig. 74: Side elevation. Fig. 75: Transverse section.

fers from other cylindrical furnaces inasmuch as its action is intermittent instead of being continuous.

The Brückner furnace has not yet been employed for blende roasting, but it is used largely for the calcination of lead and copper ores. The speed at which the cylinder is turned is varied according to the ore. In burning difficult ores it has been in some cases made as slow as four and even one and a half revolutions per hour; in chloridizing silver ore it has been run as fast as 15 revolutions per hour. Ordinarily from 2 to 3 h. p. are required. The cost of an 8·5×18·5 ft. furnace, a common size, is about $6000.

In roasting pyritous ores the charges remain in the furnace from 36 to

48 hours. The consumption of fuel is relatively low, because it is used only when and where it is actually needed. One man per shift can attend to three furnaces. The make of flue dust is high, 5% being a not uncommon figure.

The Brückner furnace is built by a number of manufacturers and in various sizes. Fraser & Chalmers (now the Allis-Chalmers Co.) construct the following:

Size.	Weight of Iron Work.	No. of Fire Brick.	No. of Red Brick.
6 ft. × 12 ft...	17,800 lb.	1,300	18,000
7 ft. × 18 ft...	30,000 lb.	1,700	20,000
8½ ft. × 18¼ ft...	52,000 lb.	2,800	25,000
8½ ft. × 28 ft...	69,000 lb.	3,300	27,000

White-Howell Furnace.—This furnace, which is quite similar to the Oxland & Hocking, used in Sardinia for the calcination of calamine, consists of a long telescopic shaped iron cylinder, made in sections to facilitate transportation, slightly inclined, supported on friction rollers and revolved be-

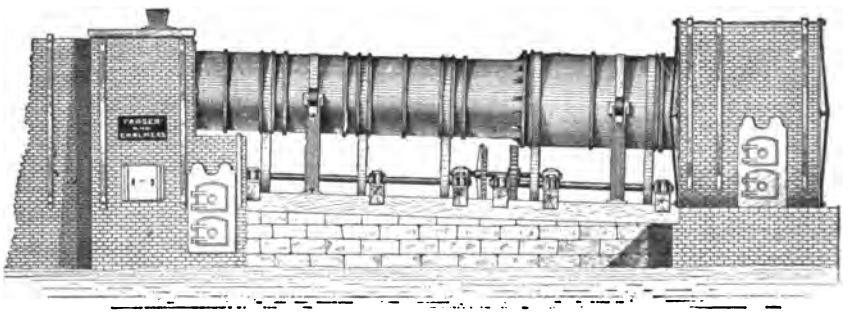


FIG. 76. WHITE-HOWELL FURNACE.

tween a stationary fire box and a flue. That portion of the cylinder nearer the fire has a larger external diameter than the part next to the flue, but it is lined with fire brick to make its internal diameter the same as that of the smaller part, which although unlined, stands the heat very well. Four projecting courses of the fire brick lining, 90° apart, assist in oxidation by raising and showering the ore through flame, which passes directly through the cylinder; for the same purpose the unlined part is provided with corresponding cast iron shelves. The furnace is fed at the upper end by means of a suitable screw feeder, and the ore makes its way automatically

toward the lower end of the furnace, where it passes out, dropping between the end of the cylinder and the fire box into a vault. Sometimes an auxiliary fire box is placed at the flue end of the furnace, as shown in the accompanying engraving, for roasting the flue dust as it passes, suspended in the air, into the dust chamber.

White-Howell furnaces are made by Fraser & Chalmers (Allis-Chalmers Co.) in the following sizes:

Diameter.	Length.	Weight of Iron Work.	Fire Brick Required.	Red Brick Required.
31 and 41 in...	23 ft.	25,500 lb.	1,900	22,000
52 and 62 in...	27 ft.	43,500 lb.	2,700	28,000

In Europe cylinders of this type are made as long as 60 ft. and of diameter as great as 7 ft.

Automatic Reverberatory Furnaces.

Under this classification are included those reverberatory furnaces in which the ore is neither stirred by hand nor mechanically, its passage through the roasting chamber being effected by gravity. The Stetefeldt furnace, used for the chloridizing roasting of silver ore, in which the ore is showered through burning gas rising in a shaft, might be considered as a furnace of this type, but usually the ore is caused to slide in a thin sheet down a series of inclined planes. This principle also finds an application in zigzag ore dryers. The Gerstenhöfer furnace, when provided and operated with an external fireplace, is an automatic reverberatory as well as a shaft furnace, but there are several furnaces of this class which do not resemble the ordinary shaft furnaces at all. Their idea is best exemplified in the Cermak-Spirek furnace, employed with great success for the roasting of quicksilver ore, and recently introduced for blende roasting at the works of the English Crown Spelter Co. at Ponte di Nossa, Bergamo, Italy.[1]

Cermak-Spirek Furnace.—This furnace is built in the form of a rectangular block, which is divided into two sections by a longitudinal wall. The latter, as well as the long outside walls, is built sufficiently thick to permit the necessary flues. The chambers on each side are filled with a rather complicated construction, the fundamental feature of which is a series of ∧ shaped fire clay tiling set across the furnace in rows, parallel with the ends of the furnace. The apex of the tiles of one row comes directly below the opening between two tiles of the next row above. At each end of

[1] V. Spirek, in The Mineral Industry, VI, 572, and X, 684.

the furnace there is a gas generator which communicates with a combustion flue in the middle longitudinal wall. From the combustion flue ports open into the chambers on each side.

The ore fed through suitable openings in the roof of the furnace distributes itself over the ∧ shaped tiling. The latter are set at such a distance that if all were covered with a sheet of ore and none were drawn off from the bottom there would be no movement. When, however, ore is drawn off from the bottom more slides down from the shelves above to take its place. It is in that manner that the passage of the ore through the furnace takes place. Practically the same thing occurs in the Gerstenhöfer furnace, and in this particular there is no great novelty in the Cermak-

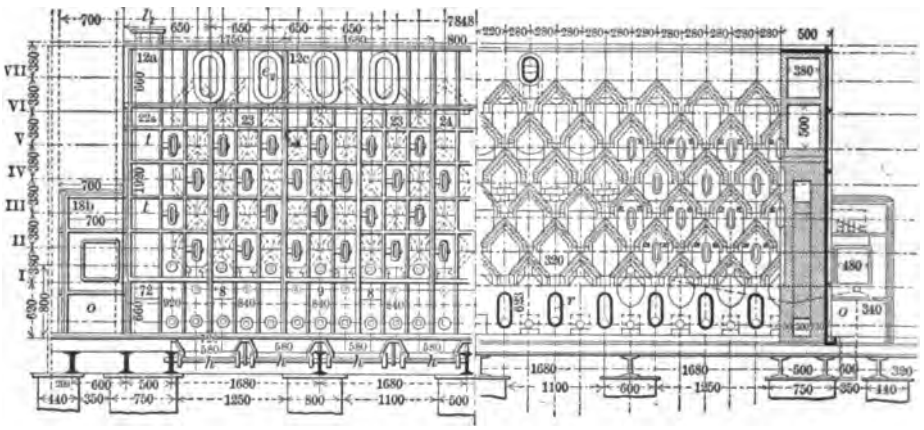


FIG. 77. CERMAK-SPIREK FURNACE.

Front elevation and longitudinal vertical section. Dimensions are in millimeters.

Spirek furnace. The original feature of the latter lies in the way the flames are caused to pass over the ore. It will be observed that the arrangement of the tiling divides each lateral chamber into a series of small rhomboidal cells, of which the axes are parallel with the ends of the furnace. The ports from the combustion flue in the middle wall communicate with the lowest tier of cells. The flames pass through those cells and enter a flue in the outside wall, from which they pass back through the next tier of cells and enter a flue in the middle wall, and so on through each tier of cells. The furnace is therefore an aggregate of a large number of multiple hearth reverberatories.

Below the lowest tier of tiles there are cast iron pipes, around which the hot ore lies until it is drawn off from the bottom of the furnace. Air is

admitted through these pipes and is thus preheated for use in the furnace while the ore is cooled so that it can be handled directly. This is the only furnace I know of in which heat is recuperated from the ore—an entirely logical procedure when it can be done conveniently as happens in this case. In the side walls of the furnace there are apertures, ordinarily closed, through which tools can be introduced to remove obstructions to the descent of the ore when necessary. It is apparent that this furnace can be used

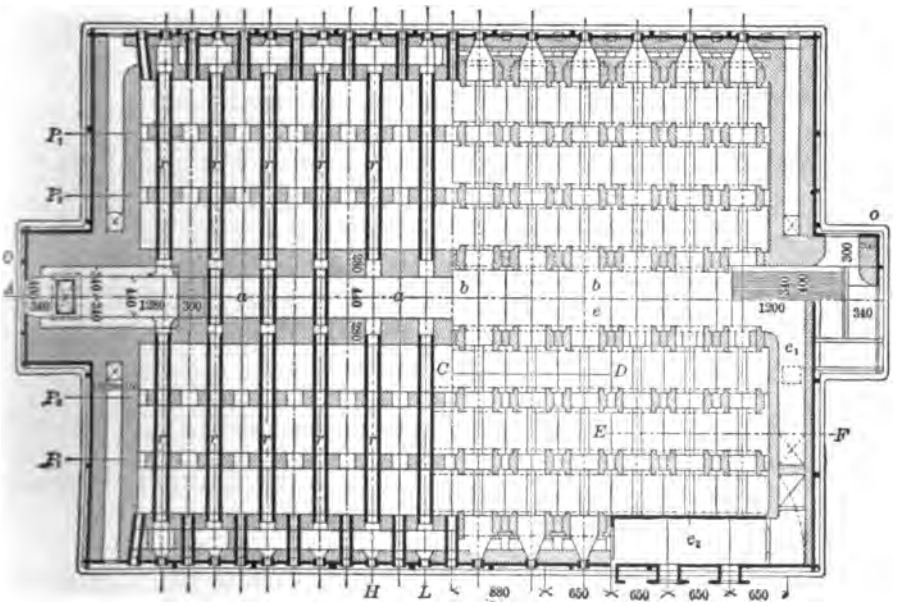


FIG. 78. CERMAK-SPIREK FURNACE.
Horizontal sections. Dimensions are in millimeters.

only with ores which run freely. Ores which sinter would give no end of trouble.

The Cermak-Spirek furnace as employed for the roasting of quicksilver ore at Monte Amiata, Italy, is illustrated in Figs. 77 and 78. There are 46 roasting cells on each side of the furnace, 11 and 12 in a row and four rows high. Each cell is 1·85 m. in length, wherefore the zigzag travel of the flames through them is $4 \times 1{\cdot}85 = 7{\cdot}40$ m. Consequently the furnace consists practically of a series of reverberatories four hearths high, with a total hearth length of 7·40 m. The air supply enters partly through the tubes *r* which communicate with the flue *a*, located directly beneath the

combustion chamber *b* and connected with it by a series of ports. Air enters also into the roasting cells through the short tubes which are shown alternating with the tubes *r* in Fig. 78. The spent ore is discharged from the furnace at a temperature below 100° C. and the air is preheated in the tubes *r* to between 300 and 600° C. The grates at the opposite ends of the furnace are fired alternately, so that the highly heated gases from one flow against the smoky gases from the other, and the flue *b* being of comparatively large sectional area the velocity of the gases in it is slow and thorough mixture of the gases and complete combustion take place. The temperature in the lowest row of roasting cells is 700 to 800° C.; in the second 500 to 600°; in the third 500°, and in the uppermost 360 to 400°.

The furnace at Ponte di Nossa is of the same general design as those at Monte Amiata, but presents certain modifications. It is square in plan and is eight cells high, with five and six in a row on each side of the furnace, while the iron pipes for preheating the air are dispensed with. This furnace has been used both for the calcination of calamine and for blende roasting. Results as to the latter have not been published, but in the former a capacity of about 300 metric tons per month, with the consumption of 8·5% of coal, has been reported.

Muffle Furnaces and Acid Kilns.[1]

The physical differences between roasting in muffle furnaces and roasting in reverberatories have been discussed in a previous chapter. Muffle furnaces are more generally used in Europe than in America, especially in Belgium and Germany, where many of the zinc smelters are situated near a good market for sulphuric acid and where the law forbids sulphurous fumes to be discharged directly into the atmosphere without previously rendering them innocuous. For those reasons reverberatory furnaces have generally been abandoned for blende roasting, muffle furnaces of the type of the Eichhorn & Liebig and the new Hasenclever (Rhenania) having been substituted for them.

The best European practice in blende roasting is to be found probably in Westphalia and Rhenish Prussia, where the manufacture of sulphuric acid in that connection has long been practiced under the auspices of the

[1] Under this classification have been included certain furnaces like the Herreshoff and Spence, which are employed chiefly in the roasting of ores for sulphuric acid manufacture without the aid of extraneous fuel. Although the grouping of them with muffle furnaces is admittedly inaccurate from the scientific standpoint, it has in view the industrial distinction between furnaces which are adapted to deliver a strong sulphurous gas suitable for acid manufacture and those which are not.

Chemische Fabrik Rhenania, of which the late Robert Hasenclever was the director and an eminent authority on this subject.[1]

The first Hasenclever-Helbig furnace was introduced as early as 1870.[2] They were subsequently installed at other works in Germany, but have now been entirely abandoned; the last of them, at a works in Upper Silesia, having been dismantled early in the decade 1890 to 1900. The Hasenclever-Helbig furnace was followed (in 1880) at the Rhenania works by the Eichhorn & Liebig;[3] this was in turn succeeded by an improved furnace designed by Hasenclever, which under his name or sometimes known as the "Rhenania," is the most popular in Germany at the present time.

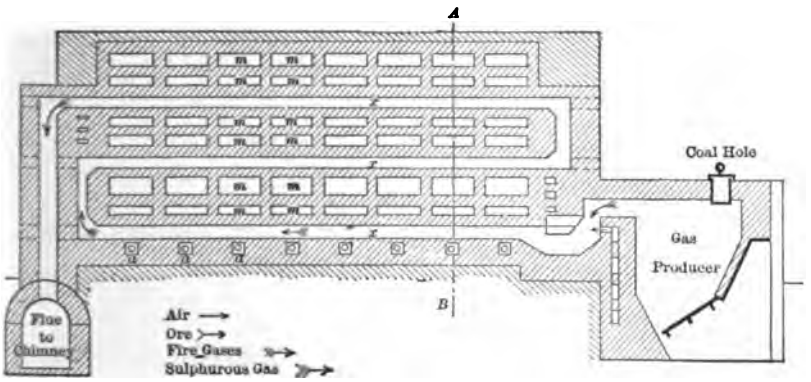


FIG. 79. EICHHORN-LIEBIG FURNACE.

Section on line C D of Fig. 80.

Only four works in the United States have yet undertaken the manufacture of sulphuric acid, and consequently there has been no general cause for the introduction of muffle furnaces. At Lasalle, Ill., the Hegeler furnace, a mechanically raked muffle type, has been in use since 1881.

Muffle furnaces are classified in the same manner as the reverberatories, i.e., as (1) raked by hand; (2) mechanically raked; (3) revolving cylinders; and (4) automatic. There is no good example of the last class, although the Hasenclever & Helbig furnace embodies its principle, the ore sliding by gravity down an inclined hearth heated from below.

[1] The Chemische Fabrik Rhenania established sulphuric acid works in connection with various zinc smelteries in Rhenish Prussia and Westphalia, including the plants at Stolberg, Dortmund, Hamborn and Oberhausen, the sulphurous gas being bought from them, although this is an arrangement which does not always operate satisfactorily, inasmuch as the processes of blende roasting and acid manufacture must necessarily be conducted harmoniously in order to secure the best results.

[2] Berg- u. Hüttenm. Ztg., 1873, p. 16; 1879, p. 176; 1886, p. 80 and p. 180.

[3] Ibid., 1883, pp. 260, 273, 413, 479, 505; 1884, p. 5; 1886, p. 201.

Raked by Hand.

Eichhorn-Liebig Furnace.—This was the first of the modern hand worked muffle furnaces. Its principle is shown so clearly in the accompanying engravings that an extended description is unnecessary. The furnace is heated by gas from a producer, the gas burning in the canals *x x x* and escaping to the chimney as shown in the longitudinal section. The air supply for the muffles enters through pipes and channels *a* under the lowermost fire canal, in order to be preheated before entering the lowermost muffles. The ore charged from the top is moved from muffle to muffle by suitable tools inserted through the doors *d*, and is finally discharged as shown. The ore remains six to eight hours in each muffle, and consequently is 36 to 48 hours in the furnace. Data as to the progress in desulphurization were

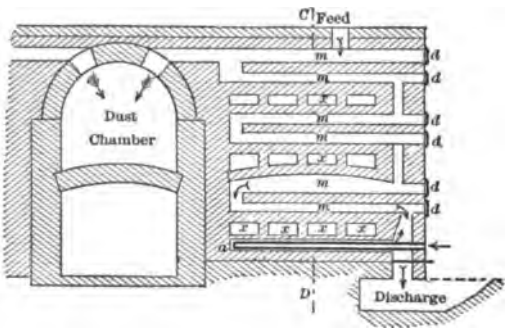


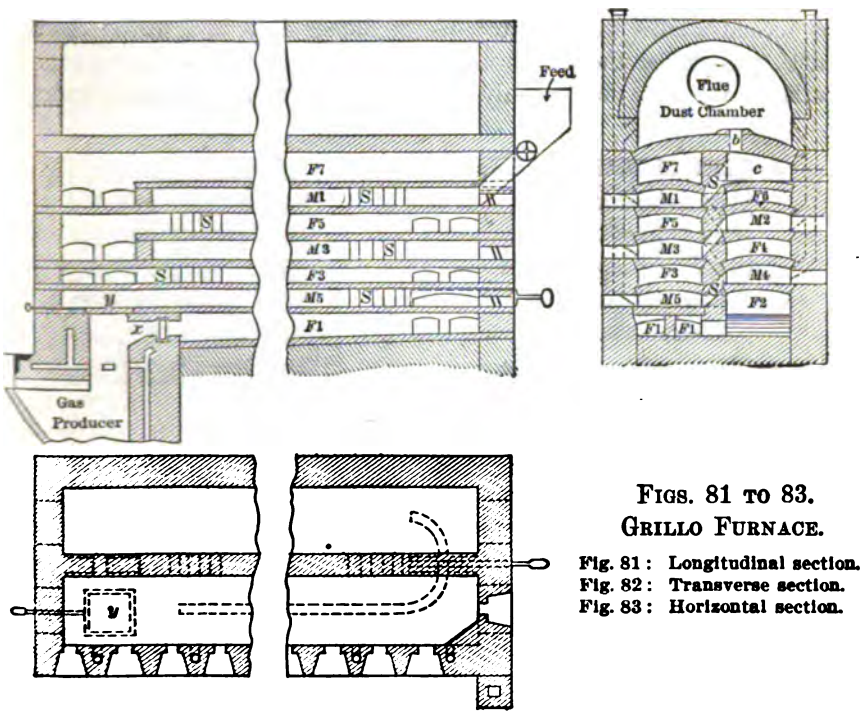
FIG. 80. EICHHORN-LIEBIG FURNACE.
Section on line A B of Fig. 79.

given in Chapter III. The roast gases contain 6 to 8% SO_2 by volume. According to Eichhorn,[1] one of these furnaces desulphurizes 4·2 to 4·5 metric tons of ore to 0·1% S per 24 hours, with a consumption of 0·8 ton of coal and the labor of four men (two per shift). These furnaces are in use at Letmathe and at Hamborn near Oberhausen, in Westphalia. At Hamborn 5000 kg. of ore are roasted in 24 hours by four men with 20 to 25% of run of mine coal (Förderkohlen). The furnaces of this type which were employed at first had eight rows of muffles or six rows (as in Figs. 79 and 80), but this multiplication proved unnecessary and the later constructions have only three or four rows (vide pp. 45 and 46).

Grillo Furnace.—The arrangement of this furnace is shown in the accompanying engraving, in which *M M* are the muffles, and *F F* the combustion

[1] Fischer's Jahresbericht, 1889, p. 322.

flues. The muffles are 9 m. long. The ore fed mechanically from the hopper, as shown, enters the muffle M, and thence passes successively through M_2, M_3, M_4 and M_5. The muffles are heated upon three sides by gas from the producer, which passes successively through the flues F_1-F_7. The sulphurous gas, passing from one muffle to the next above, through the ports S, in opposite direction to the ore, arrives finally in the canal c, whence it is led into the dust chamber and thence through a proper flue to the Glover tower. Air for combustion of the coal gas enters the canal from

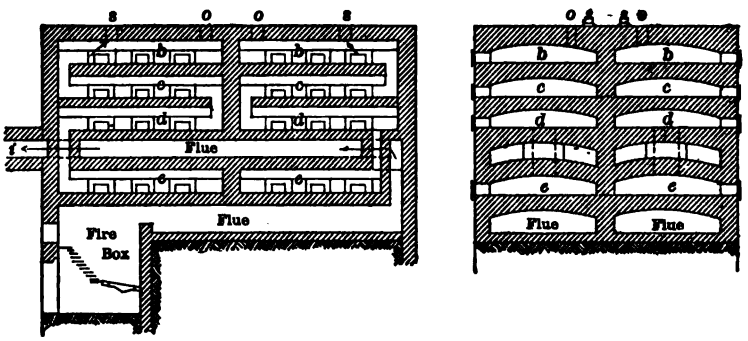


FIGS. 81 TO 83.
GRILLO FURNACE.
Fig. 81: Longitudinal section.
Fig. 82: Transverse section.
Fig. 83: Horizontal section.

the generator through ports from channels surrounding the generator in the usual manner. By removing the plate y and closing the damper x the furnace is converted into a reverberatory. This furnace was used at Hamborn-Neumühl, Westphalia, but did not prove entirely satisfactory and has been displaced there by the Eichhorn-Liebig.

Hasenclever Furnace.—Of the modern muffle furnaces, worked by hand, this is the simplest and perhaps the most efficient; at least it is the most generally used. It was built originally with three hearths, each with twelve working doors, and an ordinary fireplace, with plane or step grate, at one

end of the furnace. The flames drew first through flues under the third (lowest) hearth, then between the third and second, second and first, and finally over the first to the chimney, whereby a complete circuit was made around each muffle, as is shown in the accompanying engravings. The ore was worked along the hearths in the reverse direction to the flames in the combustion flues, while the sulphurous gas escaped through each muffle in succession to the collecting flue, or else was drawn directly from each muffle by ports in the longitudinal wall dividing two furnaces, built in pairs, as shown in Fig. 86. In furnaces subsequently built the flues between the first and second muffles were omitted, so the flames drew first under the third muffle, then under the second and finally over the first (uppermost). In the latest furnaces there is only one circuit of flues, which pass under the



FIGS. 84 AND 85. HASENCLEVER FURNACE AT GUIDOTTOHÜTTE.
Fig. 84: Longitudinal section. Fig. 85: Transverse section.

lowest muffle and back over the uppermost. These modifications were made after calorimetric determinations and practical observation had given the uniform result that the heat of combustion of the blende is greater in some parts of the furnace than that which is derived from the coal; wherefore there was nothing to be gained by putting flues under those parts of the furnace where the ore was still high in sulphur, and of course a saving was effected in thus simplifying the furnace. A further improvement consisted in dividing the furnace transversely into two parts, as shown in the drawing of the furnace at the Guidottohütte in Upper Silesia, Figs. 84 and 85, whereby its capacity was increased 30% and from 25 to 30% of labor was saved.[1] The flues pass around both sections in one circuit, or under the lowest and next lowest, as in Figs. 84 and 85, where there are four hearths.

[1] Berg- u. Hüttenm. Ztg., Sept. 29, 1893.

The Hasenclever furnace as at present constructed has hearths 12·5 m. long and 1·5 m. wide, divided into two sections each 6·25 m. long. The arch of the lowest muffle is 0·23 m. above the hearth at the middle and 0·125 m. at the sides; the upper muffles are a little higher. The outer walls of the furnace are constructed of 0·375 m. of red brick lined on the inside with 0·125 m. of fire brick; 0·375+0·125=0·5 m., or about 20 in. The interior dividing walls are of fire brick, 0·25 m. thick, when there are no channels in the longitudinal wall for the sulphurous gas. The arches and soles of the muffles and the surrounding flues are constructed of fire brick tiles of special shapes, which are carefully fitted together. Each hearth has five cast iron doors (i.e., five in each section), which slide in grooves and can be closed tightly, admitting of a careful regulation of the air supply, The draught is sometimes regulated by the aid of a manometer. The air supply is preheated before entering the muffles by passing through conduits immediately under the lower flame flues. Two furnaces are usually built in a massive, with a common parting wall between them and independent fire boxes at one end. The ore to be roasted is dried on the top of the furnace and is pushed along the first hearth by men working through the side doors; at the end of the first hearth it drops through a hole to the next and so on. These furnaces are in use, among other places, at the Münsterbusch and Birkengang works at Stolberg in Rhenish Prussia, at Dortmund and Oberhausen in Westphalia, at the Muldnerhütte at Freiberg (Saxony) and at the Guidottohütte (Chropaczow), Silesiahütte (Lipine) and Reckehütte (Rosdzin) in Upper Silesia.

In its present form the Rhenania furnace constitutes practically a block of four single furnaces, with, however, only two fireplaces. When there are three muffles the total hearth length is 18·75 m. (61·5 ft.). The hearth area per massive is 1210 sq. ft. Eight men per 24 hours are required to attend to the massive, wherefore one man per shift has to care for 302·5 sq. ft. of hearth. According to Hasenclever,[1] the newest furnaces (in Rhenish Prussia and Westphalia) roast 9400 kg. (20,723 lb.) of raw ore per 24 hours with the labor of eight men. Assuming a hearth area of 1200 sq. ft., such a capacity would be equivalent to 17·25 lb. per sq. ft. per 24 hours, one man attending to 300 sq. ft. of hearth. The coal consumption appears to range from about 25% of the weight of the raw ore in Upper Silesia, where inferior grades of coal are chiefly used, to as little as 15% in Westphalia, where fat coal of better quality is employed. In respect to quantity of ore roasted per man and per square foot of hearth the Hasenclever furnace appears to be on an equality with the reverberatory furnaces;

[1] Chemische Industrie, Jan. 15, 1899, XXII, 11, 25.

in point of coal consumption it is clearly superior. This is due improved conservation of the heat in the furnace, the exterior wall as much as 20 in. thick; to the low muffles, of which the arches acc and radiate back the heat of the burning sulphur; to the preheatin air for oxidation, although it is not done recuperatively; and to clusion of excess air to the maximum possible extent. The workin are rather small and are made so that when they are closed the leakage through them. These furnaces are frequently managed so roasting ore with about 30% S down to 1% S, the gas will cont SO_2 by volume. This means that only a little more than twice the of air which is required theoretically for the conversion of the zin ore into ZnO and the sulphur into SO_2 is admitted into the muffle furnace. In a reverberatory furnace, yielding gas as rich as 2 (which is probably a rather high figure) the quantity of excess air through it must be approximately four times what is theoretically by the ore.

Data as to Hasenclever furnaces at various works in Europe are the following paragraphs:

A massive built at Freiberg, Saxony, in 1893 comprised four sin naces of three muffles, each with hearths $6{\cdot}25 \times 1{\cdot}5$ m. The lowest mu 0·23 m. high at the center and 0·125 m. at the sides; the two upper were 0·28 m. at the center and 0·175 m. at the sides. The exterio were 0·5 m. thick; the interior dividing walls 0·25 m. There were t places to the massive. The burning gas passed first under the lowest and returned over the uppermost.

At the Guidottohütte the Hasenclever furnaces have four muf which only the two lower are heated extraneously, the two upper heated by the burning of the sulphur alone. One fireplace heats a m Eight men roast 10,000 kg. of ore per 24 hours, the consumption being 20% of the weight of the raw ore.[1]

At the Silesiahütte, Lipine, in 1895, eight men per 24 hours r 10,000 kg. of blende down to 1% S, with consumption of 25% of low coal, in a furnace similar to those at the Guidottohütte. The gas con 5% SO_2.[2]

At the Reckehütte, Rosdzin, in 1895, eight men per 24 hours roaste kg. of ore down to 1·0 to 1·5% S in a furnace of three muffles in height coal consumption was 2400 kg., or 34%. The gas contained 6% S volume.[3]

[1] Th. Dahlblom, Berg- u. Hüttenm. Ztg., 1891, p. 449.

[2] Schnabel, Handbuch der Metallkunde, II, 76.

[3] Schnabel, loc. cit.

At Stolberg, Rhenish Prussia, eight men per 24 hours roasted 9000 kg. of ore down to 0·5 to 1·0% S, except that calcareous and magnesian ores may still contain 2 to 3% S. The raw ore assays 27 to 28% S. The consumption of coal is 20% of the weight of the raw ore. A temperature of 1000° C. is attained during the roasting. The ore is crushed to 2 mm. size. Four charges are drawn per 24 hours.

At Dortmund, in 1894, the massives comprised four single furnaces, each of three muffles. The hearth dimensions were 5×1·6 m. (16·4×5·25 ft.), giving a total hearth area of 1033 sq. ft. per massive. Each hearth had five working doors. The muffles were 0·4 m. (16 in.) high in the center. Each furnace was attended by one man per shift (eight men per massive per 24 hours), who drew two charges of 500 kg. each per shift (12 hours), corresponding to 8000 kg. per massive, or 9300 kg. (20,500 lb.) of raw ore if the loss of weight in roasting were 14%. This would be equivalent to approximately 20 lb. per sq. ft., each man per shift attending to 258 sq. ft. of hearth. The consumption of coal (average of 24 furnaces in the works) was 17·7% of the weight of the roasted product, which would correspond to 15·2% of the weight of the raw ore. The roast gases contained 7% SO_2. The ore roasted (blende from Greece, Spain, the Harz and Westphalia) assayed 23 to 27% S and was burned to 0·4 to 1·5% S. It was crushed to 2·5 m. size. On the uppermost hearth it was caused to lay as much as 20 cm. thick; on the middle hearth, 13 to 15 cm.; on the lowest, about 10 cm. Each massive has two fireplaces, which are set side by side at one end of the massive. Beneath the lowest hearths of each side of the massive there are two flues, extending the entire length of the massive. The burning gas passes first through one of those flues, then returns through the other to the fire end of the furnace, then passes up through a vertical exterior flue, whence it traverses a flue above the uppermost muffle; from the last it passes to a canal leading to the chimney, which is common to a number of furnaces. The air required for oxidation of the ore is preheated by circulation through ducts beneath the flues under the lowest muffle.[1]

The Rhenania furnaces in use at Oberhausen, in 1894, had three muffles, and two furnaces were combined in a massive. The hearths were 10×2·1 m. (32·8×6·9 ft.=226 sq. ft.). The total hearth area was therefore 2×3×226 =1356 sq. ft. Each muffle had seven working doors, 1·6 m. apart, center to center, except the lowest muffles, which had only six doors, the place of the seventh being occupied by the fireplaces. The muffles were 0·35 m. high in the middle. The grates of each of the two fireplaces were 1·8×0·4 m. (5·9×1·31 ft.=7·73 sq. ft.). The lowest hearth was 0·54 m. (about 20 in.)

[1] Revue Universelle des Mines, 1894, XXV, 38.

above the level of the floor of the furnace house; the uppermost hearth was 1·54 m. (60 in.) above the level of the floor. Each furnace was attended by two men per 12 hours, or eight men per massive per 24 hours. The output per massive per 24 hours was 6000 kg. in the case of rich ore; 8000 kg. in the case of poor ore. The burning gas passes first under the lowest muffle, being divided between two flues, then passes upward through a flue at the end of the furnace and returns above the uppermost muffle. The air for oxidation of the ore is preheated by passage through ducts beneath the lowest combustion flues. The consumption of coal is 16 to 17% of the weight of the roasted ore.[1] Assuming that the capacity of the furnace was 7000 kg. and the loss of weight in roasting 14%, the equivalent in raw ore would be about 8100 kg. (17,857 lb.). Reckoning the hearth area as 1300 sq. ft. (allowing for the fireplaces) the quantity of ore roasted per 24 hours per square foot of hearth was approximately 14 lb., one man per shift having to attend to 325 sq. ft. of hearth. The consumption of coal on the basis of raw ore appears to have been but little more than 14%.

Hasenclever & Helbig Furnace.—This furnace was widely used by zinc smelters in Europe until a few years ago, when it was finally displaced by the new Hasenclever and other improved types. It was itself a development of an earlier form, dating back about 30 years, in which the ore was caused, as in the old Kerpely furnace, to descend a series of inclined shelves in a zigzag course. The same idea has found a recent application in the Cermak-Spirek furnace, previously described.

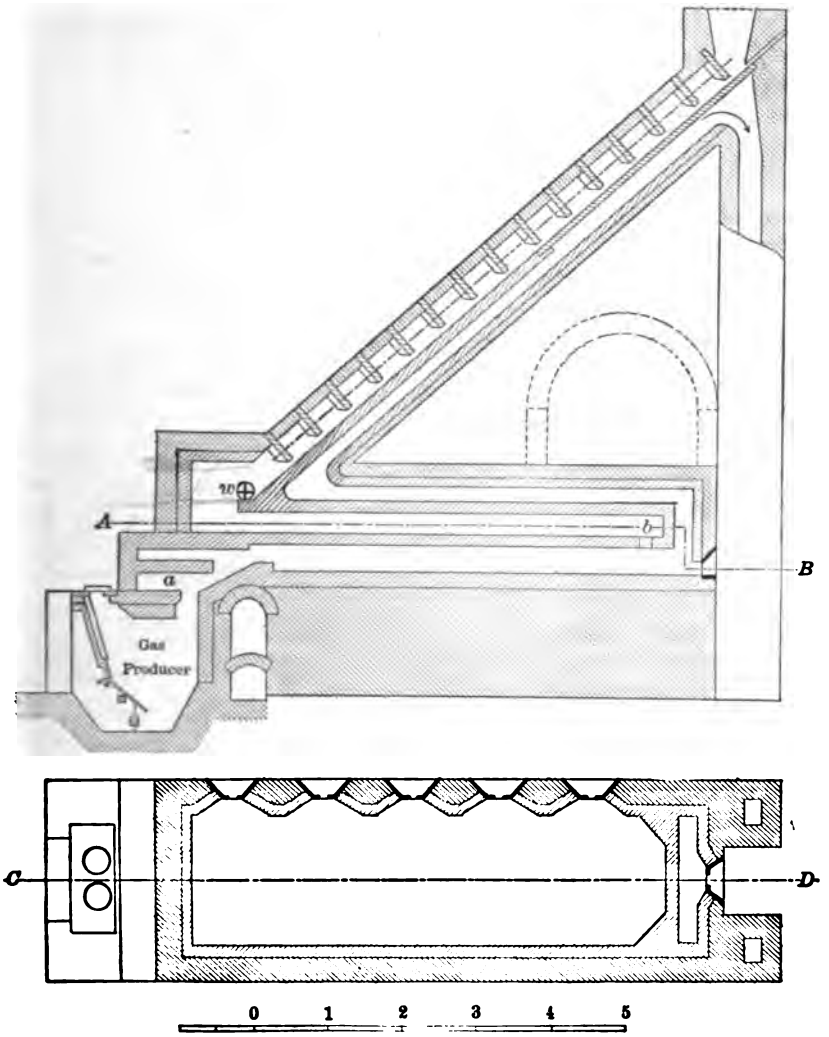
In the improved Hasenclever-Helbig furnace there was an inclined hearth sloping at an angle of 43°, from the roof of which there were projecting tiles every 0·5 m. which reached within a few centimeters of the hearth. These served to cause the ore to lie in a thin sheet on the hearth. The ore descending the inclined hearth by gravity arrived on a horizontal hearth, upon which it was worked forward by hand; it was then dropped through a hole to a lower horizontal hearth, upon which it was worked by hand back to the fire bridge, where it was discharged.

The gas from the fireplace, or generator, was burned over the lower horizontal hearth, which was consequently a simple reverberatory; it then was conducted through flues over the upper horizontal hearth, which was therefore a muffle, and under the inclined hearth to a flue leading to the chimney. The sulphur driven off on the lowest hearth was of course lost. The gas from the upper horizontal hearth escaped to the sulphuric acid department through the inclined muffle. Inasmuch as the plates hanging from the roof of the latter would prevent direct passage, ports were arranged in the side

[1] Data communicated by Mr. Ross, Revue Universelle des Mines, XXV, 47 et seq.

walls so that the gas could pass back and forth through each division in succession.

The arrangement of this furnace is illustrated in the accompanying engraving, which shows a fluted wheel, *w*, for the regular discharge of the ore



FIGS. 90 AND 91. HASENCLEVER & HELBIG FURNACE.

Fig. 90: Vertical section on line C D of Fig. 91. Fig. 91: Horizontal section on line A B of Fig. 90.

from the inclined hearth to the upper horizontal hearth. Two such furnaces were commonly united, back to back, in a block.

A furnace of the dimensions shown would roast 3000 to 4000 kg. of ore in 24 hours down to 1% S with a consumption of 25 to 50% of coal, yielding a gas from the muffles which contained 6% SO_2 by volume. At Oberhausen furnaces of this type had the following dimensions: reverberatory hearth, 1·8×5·7 m.; horizontal muffle, 1·8×7·5 m.; inclined muffle, 1·8×10 m.; gas producer, 1·8×0·6 m. In roasting ore assaying 26·44% S, at the end of the inclined hearth (temperature of red heat) its tenor in sulphur was 8·2%; at the end of the horizontal muffle (bright red), 6·2%; and at

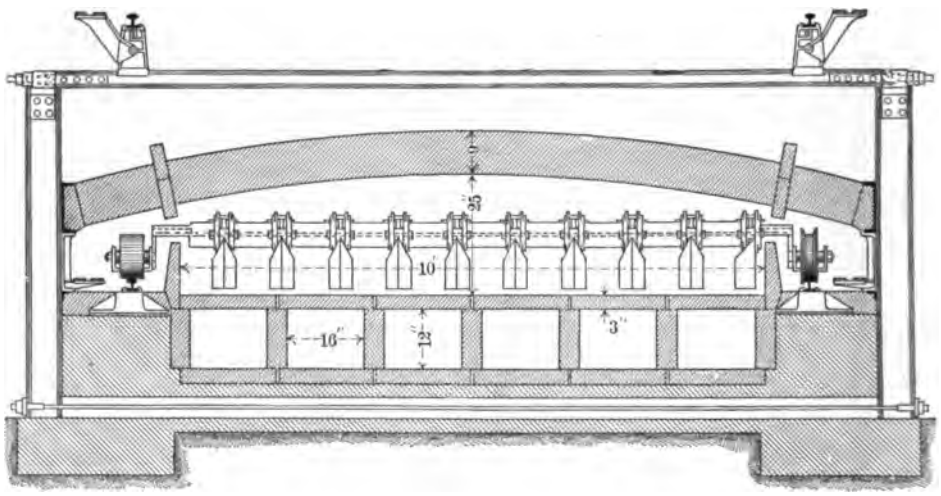


FIG. 92. BROWN MUFFLE FURNACE.
Transverse vertical section. Scale, 1 : 40.

the end of the reverberatory hearth, 0·55 to 1·3%. In 24 hours there were roasted 3500 kg. of ore with a consumption of 1050 kg. of coal and the labor of five men, the roast gases assaying 5 to 6% SO_2 by volume. Two furnaces were managed by five men per shift. The cost of roasting in these furnaces at Oberhausen was 12 fr. per 1000 kg., as compared with 7@8 fr. in the double hearth reverberatories.[1] At the Reckehütte, in Upper Silesia, the average duty per furnace was 3500 kg. of raw ore per 24 hours, roasted down to 1% S with the consumption of 2000 kg. of coal.[2] The roasting of 3500 kg. corresponds to about 17 lb. per sq. ft., the hearth area being 450 sq. ft.

[1] Mahler, Annales des Mines, VII, iii, 512 (1885).
[2] Berg- u. Hüttenm. Ztg., 1877, p. 71.

Mechanically Raked Muffle Furnaces.

Brown Muffle Furnace.—This is built in the same manner as the "straight line" furnace, previously described, but a series of longitudinal combustion flues are placed under the hearth so that the latter is heated without admixture of coal combustion products with the sulphurous gas. At the ends of the roasting chamber there are two swinging doors, the distance between them being such that one is opened and shut before the other is opened,

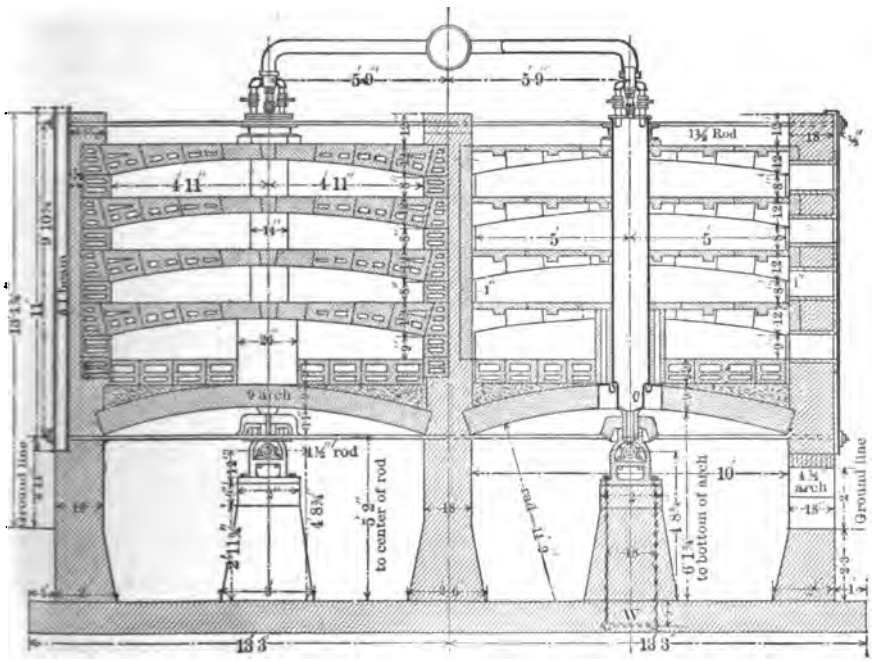


FIG. 93. FALDING FURNACE.

Transverse sections through hearths. The left-hand side is a section through the arches which support the hearth tiles; the right-hand side is a section through the columns which carry the stirring arms.

Scale, ½ in. = 1 ft.

whereby an excessive admission of air is prevented. Only one of these furnaces has been built, that one by the Bergwerks Gesellschaft G. von Giesche's Erben in Upper Silesia, and the results attained with it have not been communicated.[1] It had a hearth area of 10×150 ft., and was to be heated by producer gas. It is shown in transverse section in Fig. 92. The ironwork for such a furnace, including a 9×12 in. slide valve engine,

[1] It is in use at the Bernhardihütte.

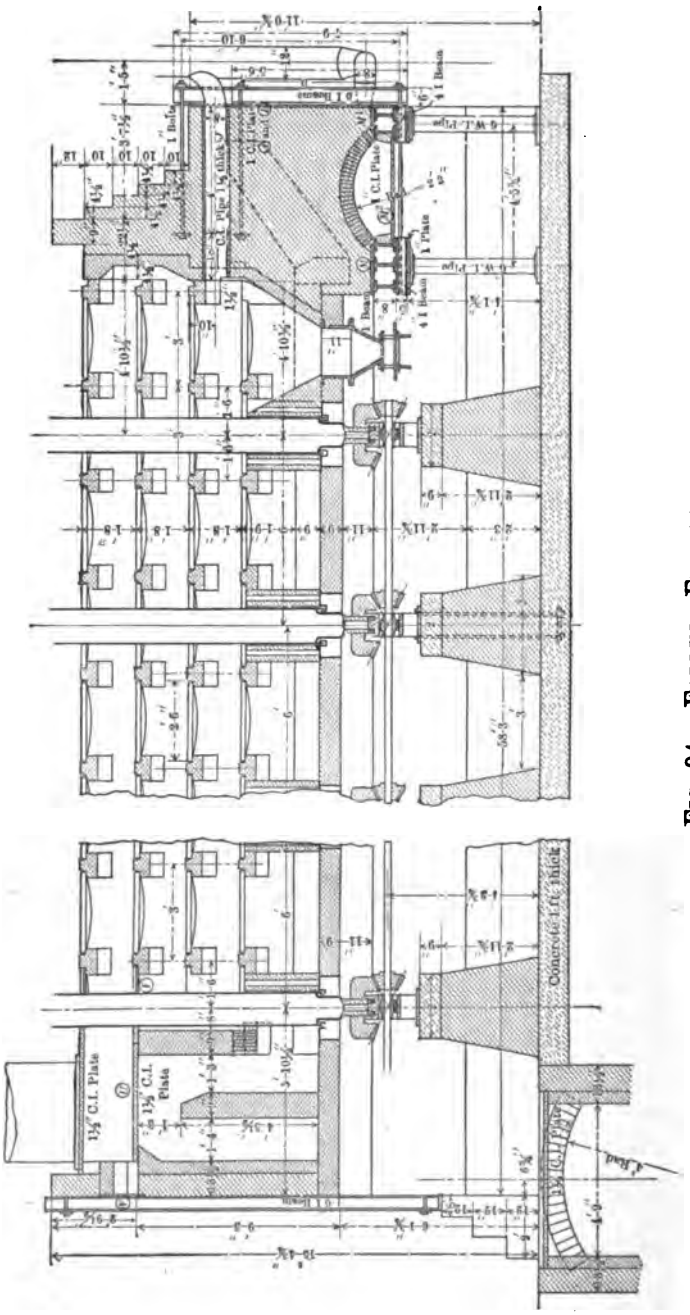


FIG. 94. FALDING FURNACE.
Longitudinal section through center of hearth.
Scale, ½ in. = 1 ft.

an ore feeder, and four stirrer carriages, weighs 73,000 lb. and in August, 1899, cost $4725 f. o. b. Chicago. The cost of the furnace erected in Kansas or Missouri, not including chimney, or the gas producer and its accessories, would probably be about $12,000.

Falding Furnace.—This furnace, of which one example has been erected

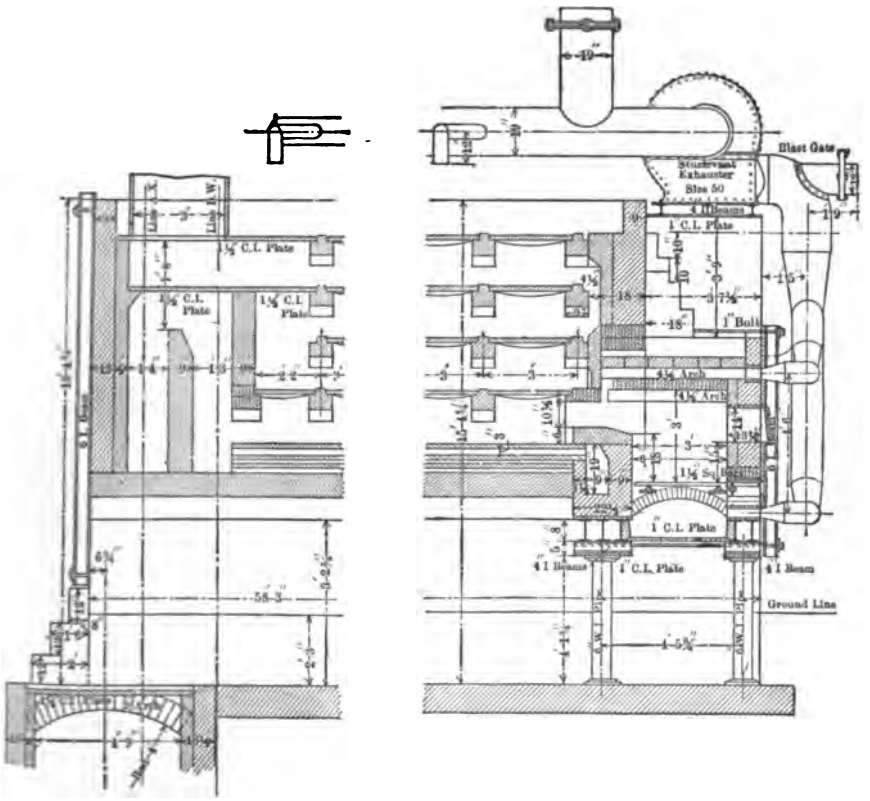


FIG. 95. FALDING FURNACE.

Longitudinal section through fire box. The furnace shown in these drawings has four fire boxes, two for each side, but this arrangement is of course capable of various modifications, either for direct or for gas firing.

Scale, $\frac{1}{8}$ in. = 1 ft.

at the works of the Mineral Point Zinc Co., at Mineral Point, Wis., is a three-hearth furnace with combustion flues under the lowest hearth. Two furnaces are united in a block, with a common middle wall. Each hearth is 10×50 ft., so the total hearth area of a double furnace is 3000 sq. ft. In

each section of the furnace there are seven vertical axes, each one bearing a stirrer arm on each hearth. The axes stand on steps, in a longitudinal tunnel under each section of the furnace, and are turned by gearing from a horizontal shaft, which can receive its power in any suitable manner.

In operation, the ore is fed on the uppermost hearth in the circle commanded by the stirrer of the first axis, which gradually sweeps it around in the direction of its travel. The circle of the second axis intersects that of the first, so that when the second stirrer comes around it engages ore, which has been left by the first, and carries it out of the reach of the first stirrer into the second circle. The circle of the third axis intersects that of the second, and so on, and thus the ore is carried forward to the end of the hearth, where it drops through a hole to the second hearth. On the latter it is moved in the same manner in the opposite direction, and on the third hearth back in the original direction to the hole through which it is discharged as a finished product.

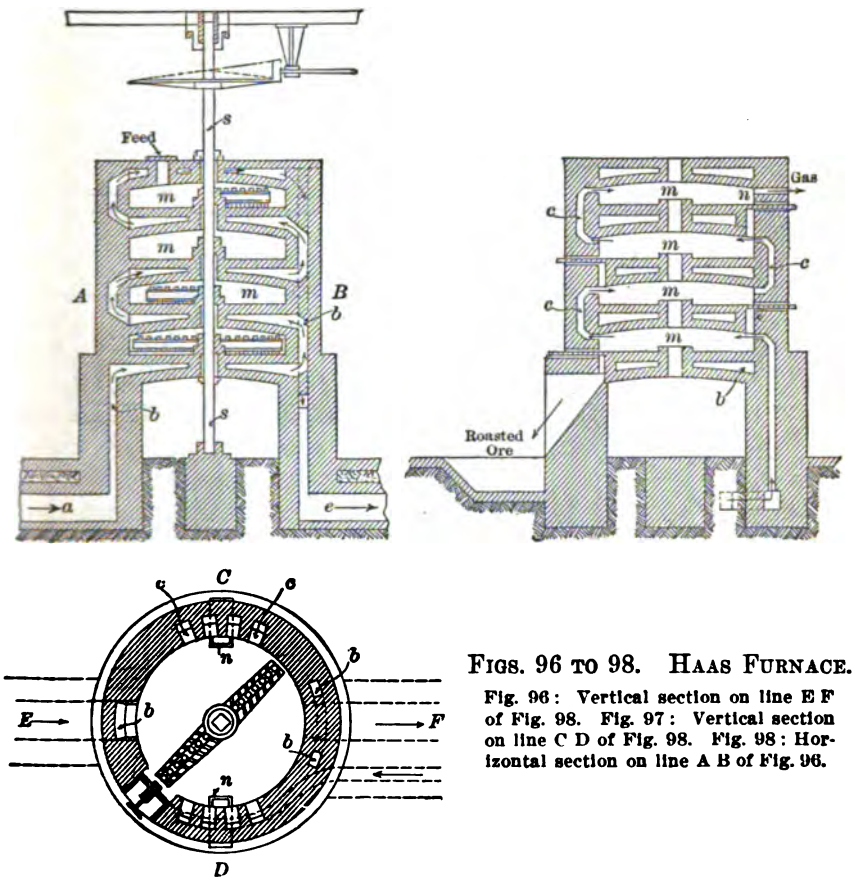
Of course the stirring arms of the various axes are set at such angles that there will be no interference during their revolutions. They are cooled by drawing air through them by means of a Sturtevant exhauster, the air being taken from the atmosphere, passing through the stirring axes and arms to the exhauster and thence being delivered into the furnace, so that the heat abstracted from the latter is returned to it, minus the loss by radiation in transit. There are doors in the side of each hearth chamber for exchange of stirrer arms, or other repairs, but ordinarily they are sealed and the air that is required for oxidation of the sulphur is forced in by the fan, so that the supply is under complete control, the furnace being perfectly tight.

In the first Falding furnace the stirring arms, of which there was one to each hearth on each axis, were turned in the same direction. The ore traveled consequently in a series of curves on one side of the hearth. This movement was found to be unsatisfactory, and in the later designs alternate axes are turned in opposite directions, which causes the ore to take a snake-like course. With the axes arranged in the latter manner each one has two arms per hearth.

The Falding furnace shows various novelties in the details of construction, especially noteworthy among which is the method of laying the hearths. The latter are formed by thin tilings which rest on transverse arches of hollow brick molded in the special shapes required. This construction is shown clearly in the accompanying engravings.

Haas Furnace.—The Haas furnace is a muffle furnace of the McDougall type, the hearths being separated by suitable flues through which the prod-

ucts of combustion from the fireplace are made to pass. The construction of this furnace is illustrated in the accompanying engraving, which shows four hearths. On each hearth there is a two-arm rabble, which is supported by and revolved by a vertical shaft in the usual manner. Each pair of muffles is connected by a flue in the wall of the furnace for the passage of the sul-



FIGS. 96 TO 98. HAAS FURNACE.

Fig. 96: Vertical section on line E F of Fig. 98. Fig. 97: Vertical section on line C D of Fig. 98. Fig. 98: Horizontal section on line A B of Fig. 96.

phurous gas, and also by an opening through which the ore may drop through to the hearth next below. The furnace is fired by producer gas, which is made to pass through the flues under the hearth of each roasting chamber, rising from one set of flues to those of the next hearth above through a duct in the side wall of the furnace. After leaving the heating flues of the topmost hearth, the gas is conveyed downward through a vertical

flue into a horizontal flue which leads into a recuperator for heating the air used for oxidation of the ore. The hot air is introduced, through a flue in the side wall, into the lowest roasting chamber, and traverses, together with the sulphurous gas which is evolved, all the roasting chambers in succession, finally escaping through a flue leading from the uppermost chamber. The ore is charged on the uppermost hearth through a suitable opening in the roof and is gradually moved forward by the blades of the rabbling arms. The vertical shaft can be turned either to the right or to the left. It has two cast iron arms in each roasting chamber, the blades of one arm

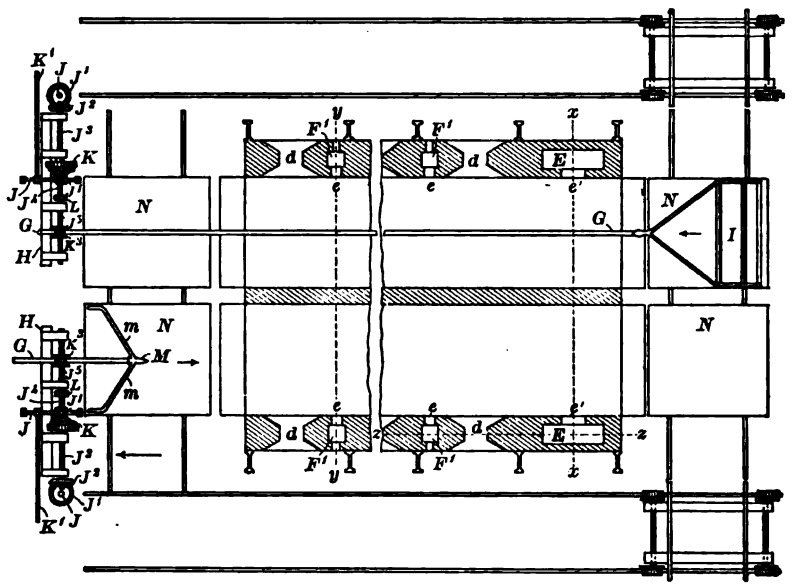


FIG. 99. HEGELER FURNACE.
Horizontal section on line *x x* of Fig. 100.

being movable and those of the other arm being fixed obliquely. By means of a lever the movable blades can be set so that the ore will be pushed either toward the center or toward the circumference of the hearth. When the ore is to be pushed from one hearth to the one next below, the blades are set to push the ore toward the circumference, whereby the ore will be shoved into the slot there, this slot being closed by a slide when so desired. These furnaces are sometimes connected with a long rectangular muffle furnace into which the ore is discharged from the lowest circular hearth.

At Oberhausen a furnace of this type is connected with a two-hearth

rectangular muffle furnace. The circular hearths are 2·5 m. in diameter, the thickness of the hearths being 0·1 m. From the lowest circular hearth the ore passes to two superimposed rectangular hearths, each 6 m. long, with four working doors, in which the roasting is completed by manual rabbling. Above the upper hearth there is a dust chamber of 24 cu. m. capacity. Two furnaces are combined in a massive. The ore roasted contains on the average 25% S. The sulphurous gas obtained has a tenor of 6 to 7% SO_2, and is used for the manufacture of sulphuric acid. The furnace is managed by one man per shift and puts through 3400 kg. of blende per 24 hours with a coal consumption of 17% of the weight of the calcined ore for heating and 5% for power. The roasted product contains on the average 1% S.[1] These furnaces were introduced at Oberhausen in

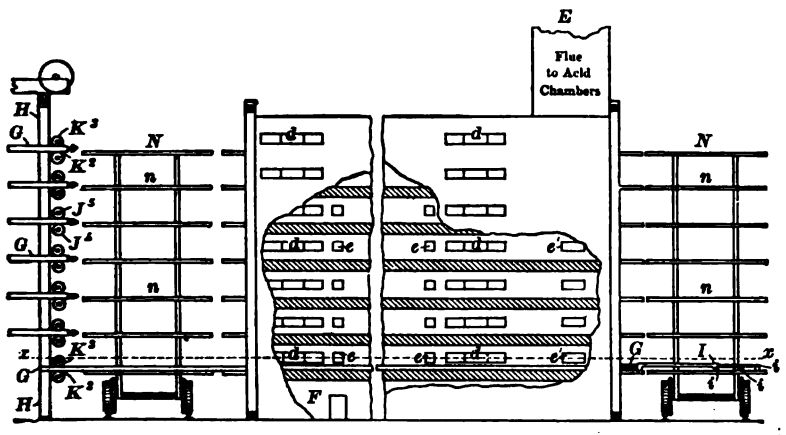


FIG. 100. HEGELER FURNACE.
Side elevation, portion of outer wall removed.

1884; in 1894 that plant comprised 24 roasting furnaces, most of which were of this type.

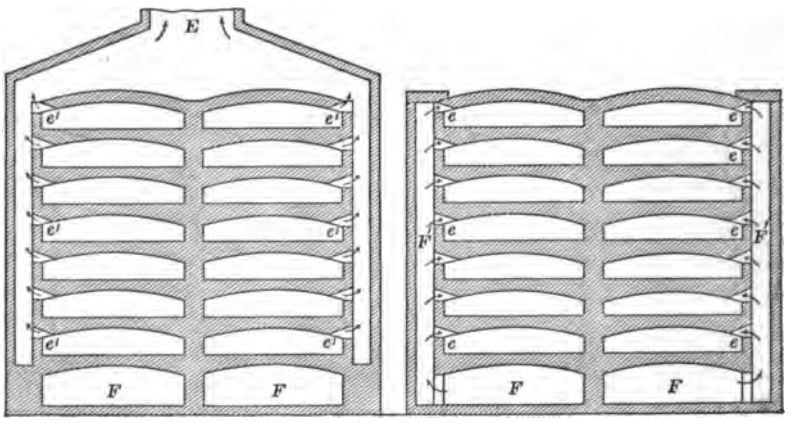
Hegeler Furnace.—The Hegeler furnace, which has been used by the Matthiessen & Hegeler Zinc Co., at Lasalle, Ill., since 1881, may be considered a modified and greatly enlarged Spence furnace, though in its present form it shows little resemblance to its prototype. It is of special interest, inasmuch as it was the first mechanical furnace to be employed successfully for blende roasting in the United States. Since the fundamental patent on this furnace expired it has been adopted also by the Illinois Zinc Co., at Peru, Ill., and the Standard Acid Co., of Iola, Kan., which, while

[1] German patent No. 23,081; Berg- u. Hüttenm. Ztg., 1884; Revue Universelle des Mines, 1894, p. 38.

preserving the general features of its construction, have altered the number and dimensions of the hearths and the method of moving the rakes.

The Hegeler furnaces at Lasalle are seven hearths high. The lower hearths are heated by gas burned in the flues beneath them, air for combustion of the gas being admitted through ports arranged at proper intervals along the flues. Air for oxidation of the sulphur is admitted to the roasting chambers in a similar manner. The Hegeler furnace has undergone important modifications since it was originally designed, but the accompanying illustrations, which are taken from the patent specifications dated August 12, 1884,[1] will serve as a basis for a description of the furnace now in use.

The furnaces shown therein are 50 ft. long and 12 ft. wide, with muffles



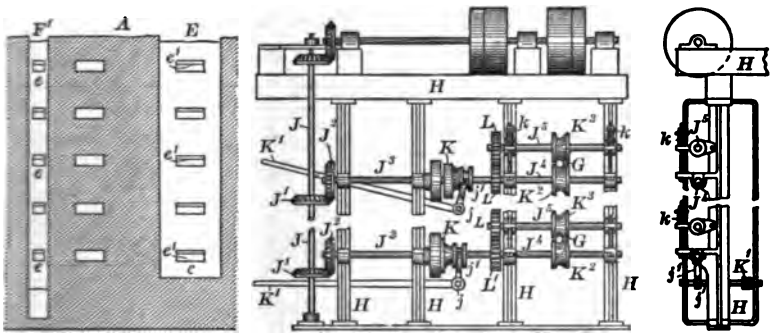
FIGS. 101 AND 102. HEGELER FURNACE.
Transverse vertical sections.

on each side of the middle longitudinal wall. The hearths are made of one course of arched fire brick 4·5 in. thick, resting on skewbacks in the central and outside walls. The muffles thus formed are 7·5 in. high and 4·5 ft. wide; they open at each end of the furnace, where there is a corresponding iron platform supported by a movable framework or car. At regular intervals in the side walls of each muffle there are openings *d*, ordinarily closed, through which tools may be inserted. Each muffle also has an opening, e^1, leading into a main flue for the passage of the sulphurous gas. Flues *F* under the lowest muffle on each side convey hot air, which is led into the muffles through branch flues F^1 and ports *e*. One or more of these branch flues F^1 are used in the roasting process, depending on the character of the

[1] E. C. Hegeler, United States Patent No. 303,531.

ore to be roasted, the others being stopped off in the meanwhile. The supply of hot air to each muffle is regulated by enlarging or contracting the ports *e*, which is effected by means of small openings through the outer wall into the branch flues F^1 opposite to the ports, which are ordinarily kept closed.

At each end of the furnace there is a frame *H*, supporting the rake rods, which is set at a sufficient distance from the end of the furnace to allow room for the car *N*. The frame *H* has a series of supports corresponding to each muffle and a series of rods *G* resting on the supports in such a position that they can be thrust lengthwise through the furnace. The frame also has mechanism for thrusting in the rods and drawing them out. This mechanism consists of a shaft *J* driven in any convenient manner and arranged so as to be operated in both directions. Shafts J^3, which are driven



FIGS. 103 TO 105. HEGELER FURNACE.

Fig. 103: Vertical longitudinal section on line *z z* of Fig. 99. Fig. 104: Front elevation of rake operating mechanism. Fig. 105: Side elevation of rake operating mechanism.

by miter gearings, are connected to shafts J^4 by friction couplings *K* whereby they can be disengaged by means of the lever K^1. To shafts J^4 grooved pulleys K^2 are attached and above shafts J^4 are shorter shafts J^5, which have grooved pulleys K^3. The rod *G* is held fast between the pulleys K^2 and K^3 by springs *k*, arranged to press the shaft of pulley K^3 toward the other, thus causing the pulleys to grip the rods. The shafts of the grooved pulleys are connected by gears *L* so that when the coupling *K* is connected the motion of shaft J^3 is communicated to shafts J^4 and J^5 and the pulleys can be operated in one direction to thrust the rod *G* into the muffle, or in the other direction to draw it out. A set of the pulleys, K^2 and K^3, is employed in connection with each of the rods *G*, so arranged as to be operated separately.

The cars N have a series of platforms n corresponding in number and height with the hearths of the muffles; they are used for the purpose of holding and transporting the rakes I, which are employed for stirring the ore. In later forms of the Hegeler furnace these cars were abandoned in favor of a permanent swinging framework, to be described subsequently, which obviates the necessity of transporting the rakes around the furnace.

The form of the rake and the method of connecting it with the rods G is shown so clearly in the accompanying engraving that no special description is required, save to remark that the sides are made of flat iron so arranged at the forward end as to scrape the sides of the muffle and prevent the ore from packing there. The design of the rake is such as to insure that the ore will be plowed up from the hearth and building up of the latter will be prevented. The travel of the ore is governed by the length of the rake and the number of triangular transverse scrapers. The rake is made almost the full width of the hearth and is about 6 ft. long. In construction it is

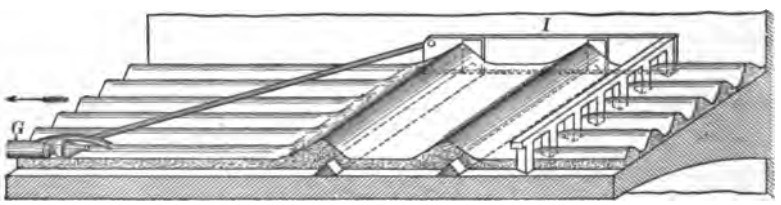


FIG. 106. HEGELER FURNACE.
Perspective view of portion of hearth, showing rake.

very massive. For a 50×4·5 ft. furnace it weighs about 550 lb. One rake is employed for each pair of muffles on the same level; when not in use they are left on the platforms n of one of the cars to cool, remaining in proper position for passage through the muffle again. The rods G are provided with a barbed head corresponding with a suitable device in the rake for attaching and detaching the rakes to and from the rods. The furnace shown in Figs. 99 and 100 has drawing rods only at one end, there being a rod for each muffle. Consequently the ore could be moved only in one direction and each muffle was practically a separate furnace. After the rakes had been once drawn through it was necessary to tram them around to the opposite end of the furnace to put them in position for another passage. The clumsiness of this arrangement is of course obvious.

In the modified form of the furnace shown in longitudinal section in Fig. 107 the ore is charged through a hopper a and after passing along the upper-

most hearth falls through the opening *b* to the next hearth below, traveling thus over each hearth in alternate directions. In this furnace the sulphurous gas passes through each muffle in opposite direction to the ore, finally escaping from the uppermost muffle into the gas main. The rake rods *G* are alternated in the respective frames so as to correspond on each side with every other muffle, instead of with every muffle as described above, and inasmuch as the ore is made to move in opposite directions a system of rods is required at each end of the furnace. This system was naturally of greater advantage than the one first described, although the necessity of a rod system at each end required of course a much greater ground space. The rakes had to be trammed around the furnace, however, as before.

The furnaces illustrated in the accompanying engravings do not show com-

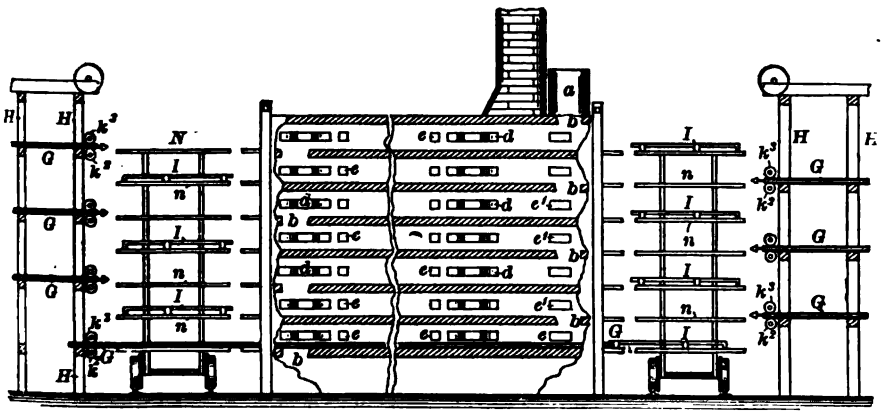


FIG. 107. HEGELER FURNACE.
Longitudinal, vertical section.

bustion flues under the hearths. In the later furnaces, and those in use at present, combustion flues are provided, the 40 ft. and 50 ft. furnaces having such flues under the sixth and seventh hearths, while the 75 ft. furnace has in addition a flue under the fifth hearth. The present furnaces are built very solidly of brick with walls 13 in. thick, bound together by tie rods and buckstaves in the usual manner. The top of the furnace is covered with a corrugated iron plate, on which the ore is dumped midway between the ends. From this point it is dragged slowly to one end on one side of the furnace and to the opposite end on the other side by means of a conveyor, having fingers working in the corrugations of the plates. By this device the ore is charged automatically into the furnace, having been

dried in the meanwhile. Having been dropped on the first or uppermost hearth it is raked slowly forward to the hole through which it drops on the second hearth, and so on, to the discharge from the lowest hearth.

The furnace is closed at the ends by heavy iron doors swinging on hinges, which are balanced by counter weights so as to open and close easily after passage of the rakes. Some air enters the furnace at the end of each hearth during the stirring of the charge, but the chief supply is obtained through ports at the side of the hearth, connecting with channels in the side walls. The air for the combustion of the gas is drawn from the same channels.

At each end of the furnace as now designed there is a carriage to receive the rakes, which is essentially a steel framework of skeleton platforms, corresponding to the hearths of the furnace, swinging around a heavy post. The rake rods, which are steel bars 2 in. sq., are operated in substantially the same manner as described for the older furnaces. The present method of

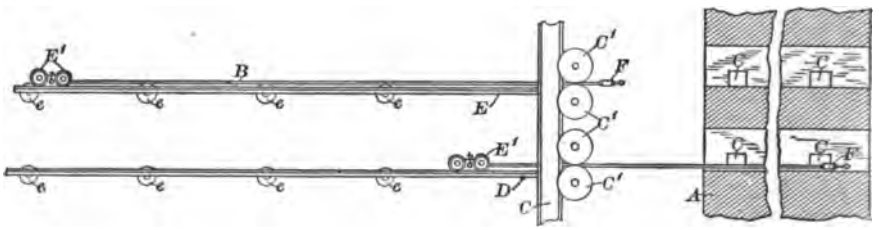
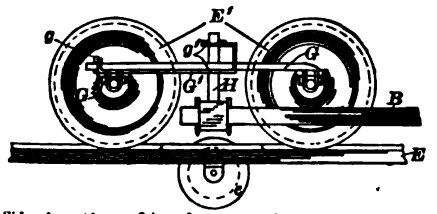


FIG. 108. HEGELER FURNACE.
Method of supporting rake rods.

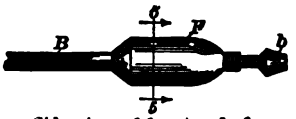
raking the ore is as follows: Assuming that the charge in the left-hand side of the furnace is to be rabbled, the carriages with the rakes are put in position at the ends of the furnace from which they are to be drawn. Beginning with the first hearth, the door at the opposite end is lifted and the rod is passed in. Having issued through the opposite door the rake is coupled on by means of the ball and socket joint, and at signal from the man at that end of the furnace, the machinery at the other end is reversed and the rake is dragged through the muffle, being received on the corresponding carriage at the opposite end. In raking the next lower hearth of course the rake travels in the opposite direction. When the raking of one side of the furnace is completed the carriage at each end supporting the rakes is swung around 180°, so that it comes opposite the doors of the hearths of the other side, when the raking is proceeded with in precisely the same manner. Thus the carriage for each hearth travels in circuit through the left- and right-hand side. Of course the ore moves in opposite

directions in the two sides. At each end of the furnace there is a platform car on a track on which the man connecting and disconnecting the rakes stands when the upper hearths are being worked.

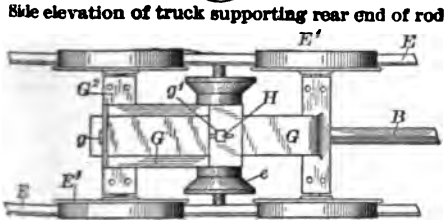
In thrusting the rods through the muffles it used to be necessary to employ a detachable guide *M* with arms *m* (see Fig. 99) to keep it in line and prevent it from being deflected to one side and gouging the brick wall of the muffle and perhaps sticking fast in one of the cleaning holes. An improvement on this and also in the method of supporting the rake rods was recently introduced by Julius W. Hegeler,[1] it having been found that with the longer furnaces the rods became so bent and warped by the heat that it was difficult to keep them in place in their supports or make them work easily



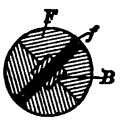
Side elevation of truck supporting rear end of rod



Side view of front end of rod provided with shoe



Plan of truck



Transverse section of shoe on line 5-5

FIGS. 109 TO 112. HEGELER FURNACE.
Details of rake rods.

therein, while the guide above referred to caused a loss of considerable time in attaching and detaching it.

In the improved device the front end of each rod is provided with an arrowhead *b* for connecting it with the rake. A cast iron shoe *F*, just back of the arrowhead, prevents it from catching in the cleaning holes and from being worn by sliding on the hearth. Instead of permitting the rear ends to drag in troughs formed by angle bars, tracks *E* are now arranged on which are placed trucks E^1, supporting the rear ends of the rods. A series of flanged rollers *e* between the track rails support and keep the rods in place, so that they may be worked easily. The truck has a T plate *G*

[1] United States patent No. 592,006, Oct. 19, 1897.

mounted on the front axle and on the rear axle a plate G^1, of which the front end is bent upwardly and backwardly, as shown in Fig. 109. There is an opening in the upturned part through which the T plate G can pass. There is also a plate G^2 on the rear axle with an upturned flange g, having a similar opening for a like purpose. Holes are provided in these plates at g^1 for a hanger H, by which the rear end of the rod B is connected with the truck below the axles. The shoe F is made in halves secured together upon the rod by rivets f. It is made with a bevel at the front end and is of sufficient length as regards the cleaning openings to pass them if in contact with the sides of the muffle without being caught thereby. Because of its thickness and of its being made of cast iron, the shoe is not likely to be bent or warped by the heat, and it not only keeps the forward end of the rod straight, but tends to keep it in a straight course through the furnace, so that ordinarily it will not be deflected toward either side.

The crew for each furnace consists of four men per shift, two at each end, of whom one is at the levers and the other attends to the rakes. The total time required for coupling a rake to the drawing rod and pulling it through the muffle is two minutes. The actual time in the muffle is 1·5 minutes, The rods are, of course, in the furnace a little longer, say about four minutes at each raking, during which time they become red hot on the edges. Before either the rake or rod is required to enter the furnace again, however, it is fairly cold. In 1899 at Lasalle there were two furnaces 40×4·5 ft. (hearth dimensions), one of 50×4·5 ft., and one of 75×6 ft. The ore charged into these furnaces is crushed to pass a 2 mm. round hole. The 40 ft. furnaces roast 18 tons of ore per day, producing 15 tons of roasted ore. The 50 ft. furnace delivers 20 tons from 24 tons of raw ore. The 75 ft. furnace roasts 48 tons of ore, yielding 40 tons of product. The raw ore contains 29 to 30% S, and the roasted ore 1·2 to 1·25%. The maximum desulphurization obtainable with the same ore in the laboratory is about 0·6%, tests varying from 0·45 to 0·90%. One man per shift attends to the gas producers for three furnaces. The consumption of coal is approximately 20% of the weight of the ore roasted. The roast gases, which contain about 4·75% by volume of sulphurous acid, are conducted to lead chambers, where they are utilized for the manufacture of sulphuric acid.

Herreshoff Furnace.—This furnace, which is an improvement of the old McDougall type, has been very successful in burning pyrites for sulphuric acid manufacture. It consists of a cylindrical shell of ¼ in. steel, 10 ft. in diameter, lined with 8 in. of red brick. The horizontal bottom of the cylinder and four arches of fire brick, 4·5 in. thick, constitute the roasting hearths. Passing through the latter there is a central vertical shaft, driven

from below, which carries the stirring arms, two per hearth. The shaft is hollow, 14 in. in diameter. Between each hearth there is a slot, 4 in. wide and 5 in. high, which passes through the shaft; these are to receive the ends of the stirring arms. The latter are held by a self-locking joint, their own weight, 100 lb., holding them in position. To remove an arm it is lifted about 3 in. at the outer end and can then be pulled out. The operation is so simple that an arm can be removed and a new one substituted in about

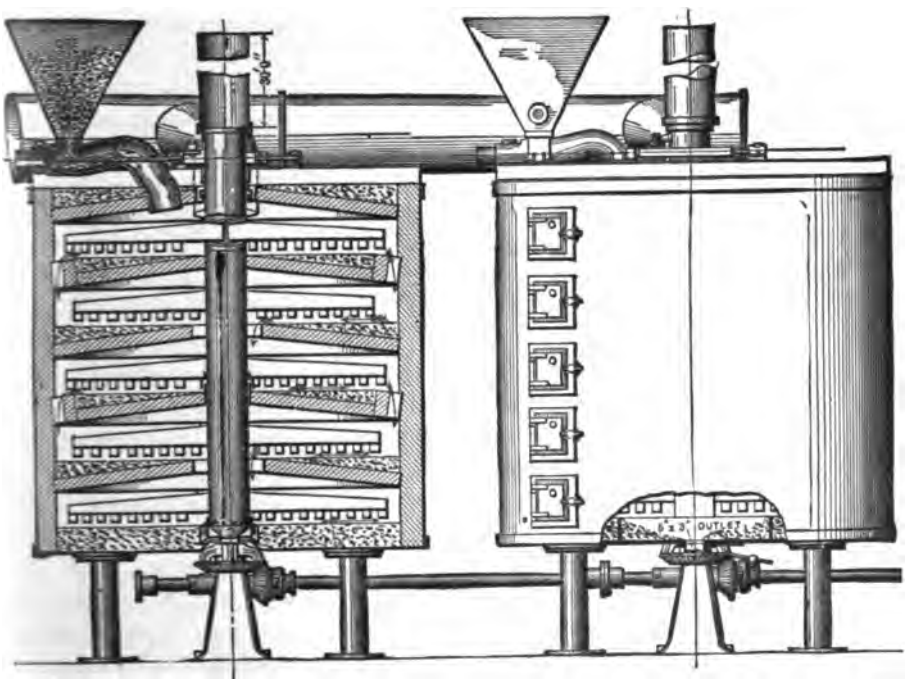


FIG. 113. HERRESHOFF FURNACE.
Vertical section and elevation.

one minute. This was one of the great improvements in the design of this furnace.

The operation of the Herreshoff furnace will be understood from the accompanying engraving without an extended description. The ore is fed mechanically from a hopper to the uppermost hearth by means of a piston working in a horizontal cylinder; the piston makes two strokes per minute. The ore dropped near the center of the first hearth is pushed out to the periphery of the latter, where it drops through holes to the next hearth

below. It is then worked toward the center of the second hearth, where it drops through a hole around the revolving shaft to the third hearth, and so on. The plows of the stirring arms are set so as to effect the proper movement. The area of the openings in the hearths is sufficient to permit free passage of the sulphurous gas, which travels in the opposite direction to the ore. The ore is finally discharged through a 5×3 in. outlet at the circumference of the lowest hearth.

The central shaft makes one revolution in two minutes. In passing through those hearths whereof the holes are at the periphery, the joint is sealed by the ore itself in the simple trap, which will be readily understood from the accompanying engraving. The shaft is cooled by drawing air through it, for which purpose a 30 ft. chimney is placed above it.[1] The stirring arms, rectangular in cross-section, are also hollow. That portion of them in which the greatest strength is required, as well as the vertical shaft, are kept below red heat. The upper surface of the arched hearths is not built up horizontal, but is leveled with sand, ashes, or ore itself.

The hearth area of the Herreshoff furnace is about 285 sq. ft. It burns in 24 hours 7000 lb. of pyrites assaying 44% S down to between 2·5 and 3·5% S. It is said that one man per shift can attend to 36 furnaces.[2]

A muffle form of this furnace has been used for roasting pyrrhotite.

O'Brien Furnace.—This is quite similar to the Herreshoff, and like the latter has been used very successfully in burning pyrites. The chief differences are in the central vertical shaft and the stirring arms. The former is made tapering, being largest at the upper end, and is mounted at the lower end upon ball bearings. The tapering form of the shaft permits it to be easily lifted out of the furnace, the stirring arms being of course detached. The stirring arms pass through holes in the central shaft, but their principal support is a long bolt which passes completely through two opposite arms, the latter being secured by nuts at the ends. By removing a nut, the arm secured by it can be pulled out horizontally. The central shaft is open at the top and the sulphur gases are taken off through a ring of apertures in the top of the furnace around it. The ore is fed by means of a screw, which is geared directly with the main driving shaft by means of an eccentric and pawl lever, the feed being adjustable without having to go to the top of the furnace.[3]

Pearce Furnace.[4]—The Pearce turret furnace, which has already been

[1] In burning pyrites fines without fuel more heat than necessary is developed, wherefore it has been found desirable to cool the furnace; hence the thin outside walls.

[2] The Mineral Industry, VI, 236.

[3] United States patent, No. 673,174, April 30, 1901.

[4] Built by the Stearns-Roger Mfg. Co., of Denver, Colo.

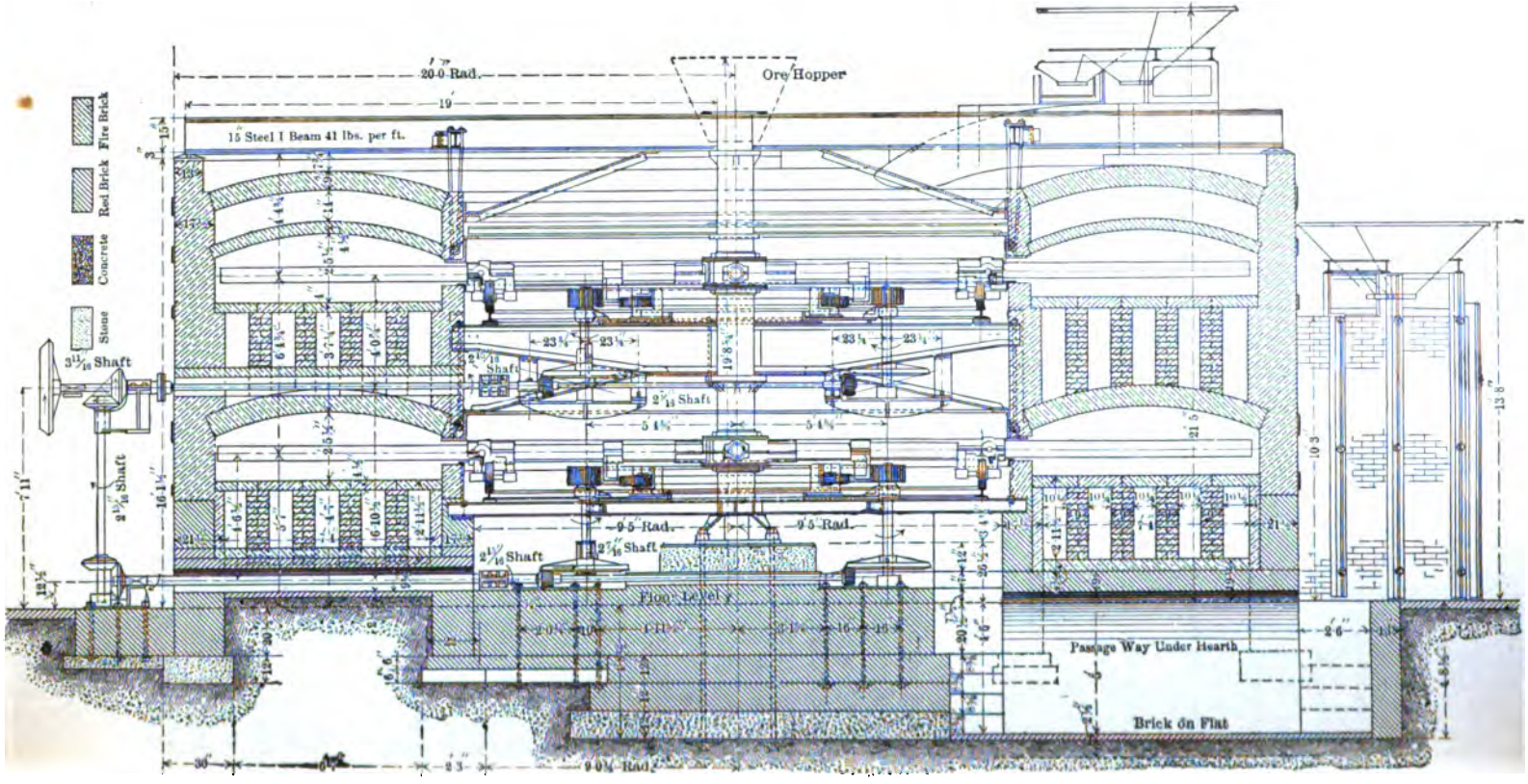


FIG. 114. PEARCE MUFFLE FURNACE WITH TWO HEARTHS.

described under the head of "Reverberatories," is easily converted into a muffle furnace by placing combustion flues under the hearths. One so arranged, which was built recently for the Pennsylvania Salt Mfg. Co., at Natrona, Pa., is shown in the accompanying engraving. This furnace has a hearth area of 1475 sq. ft. and grate area of 90 sq. ft. In its construction there were used 135,000 lb. of iron work (costing $12,000 f. o. b. Colorado), 50,000 standard fire brick, 15,000 special fire brick, 50,000 red brick, 600 hearth tiles, 66,000 lb. of fire clay, 65 bbls. of lime and 1600 bushels of sand. The large number of fire brick was due to the fact that both the side walls and entire arch were built of that material.

Rhenania Mechanical Furnace.[1]—This is an adaptation of the new Hasenclever furnace to raking by rabbles, drawn by an endless chain after the fashion of the O'Harra furnace. So far as I am aware, this furnace has not been put into practical use. As shown in the drawing accompanying the patent specification, it is a four-hearth furnace with a combustion flue under the lowest hearth and over the topmost. In order to keep the end of the furnace clear, the products of combustion have to be led from the lower flue, at the end of the latter, to the upper flue in which they are to travel in the opposite direction to the chimney, by means of vertical ducts, one on each side of the furnace and outside of the walls of the latter. At the other end of the upper flue connection is made with the chimney in a similar manner. The upper flue stops short of the feed end of the furnace, which gives opportunity to take off the sulphurous gas into a flue extending a short distance over the upper combustion flue and thence into the main leading to the Glover tower. In this manner the ends of the furnace are left clear for the entrance of the chains, which pass over sprockets supported, tautened and driven in a suitable manner. Indicating the point where the ore enters the furnace by A, the chains pass along the first hearth, over sprockets, back over the second hearth, over sprockets, back over the third hearth, over sprockets, back over the fourth hearth and over two sets of sprockets to A. The rakes are carried between the chains, which run on castings at each side of the hearths. The ends of the hearths are closed by swinging doors.

Spence Furnace.—The Spence furnace is a mechanically raked shelf burner, which may be built either as a reverberatory or a muffle furnace, or if the ore be self-burning, like pyrites, it may have no firing at all, when it becomes practically a mechanical Malétra kiln. This furnace has been

[1] Patented by the Chemische Fabrik Rhenania of Aachen, German patent No. 61,043, April 17, 1891. Although this furnace has not been used practically, a brief description is of interest in indicating the ideas of the engineers identified with Herr Hasenclever for converting the Hasenclever furnace into a mechanically raked form.

very successful in burning pyrites fines, for which purpose a large number of them are in use in the United States The original patents having long ago expired, any one is free to build them and consequently those in use show numerous variations in details. The general principles are shown, however, in the accompanying engravings of the Hammond-Spence.

The Spence furnace is now commonly built with four hearths, but it may have five, six, or even seven hearths. The latter may be brick arches sprung between the side walls, but usually are of flat, rebated tiling laid horizontally across the furnace. The courses of the ore and products of combustion are shown clearly in the accompanying engravings. The movement of the ore is effected by a rake on each hearth, which is pushed into the furnace and then drawn backward. The rakes sometimes run on iron

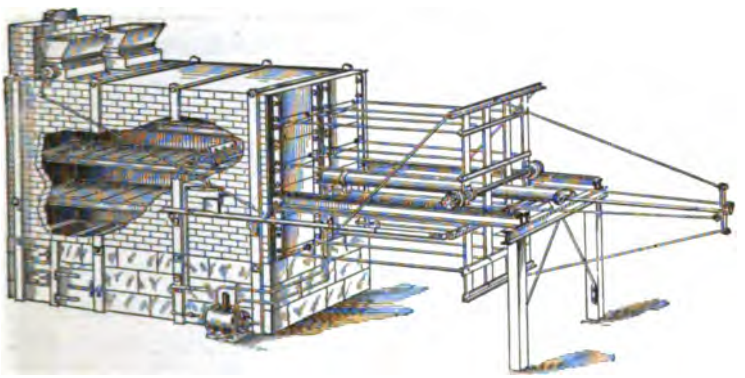


FIG. 115. HAMMOND-SPENCE FURNACE.

rails at the sides of the hearths, but the rails give much trouble, and in some furnaces have been replaced by very hard brick tiles. Both steam and hydraulic power are used to effect the movement, and the mechanism is so arranged as to work automatically. The rakes are provided with trips so that they will act only with the course of the ore and not against it. The feed may be regulated by a fluted wheel, or any other suitable device, which is connected with the rake mechanism. When not in motion the rakes rest on the portions of the hearth beyond the holes through which the ore is dropped. The rakes being constantly in the furnace suffer a good deal from the intense heat; this is one of the drawbacks of the Spence furnace. However, it is said that with ordinary care a rake will last two years and can be replaced in 10 minutes, which is certainly not a very serious charge. It is apparent that the length of hearth is limited to that at which the rake

rods can be kept straight. The furnace shown in Fig. 116, which has rake rods 16 ft. 9 in. long, making a stroke of 12 ft. 10 in., has only about 10 ft. length of effective roasting hearth. If the hearth be laid with flat tiling its width is limited to the span at which the safety of the tile can be relied upon. This is about five feet. Spence furnaces are usually built in blocks of two, with a common middle wall. A double four-hearth furnace, of effective roasting surface of only 10×5 ft. per hearth, has a total hearth area of 10×5×4×2=400 sq. ft. One man per shift can attend to three double furnaces; when there is no firing to be done one man per shift has

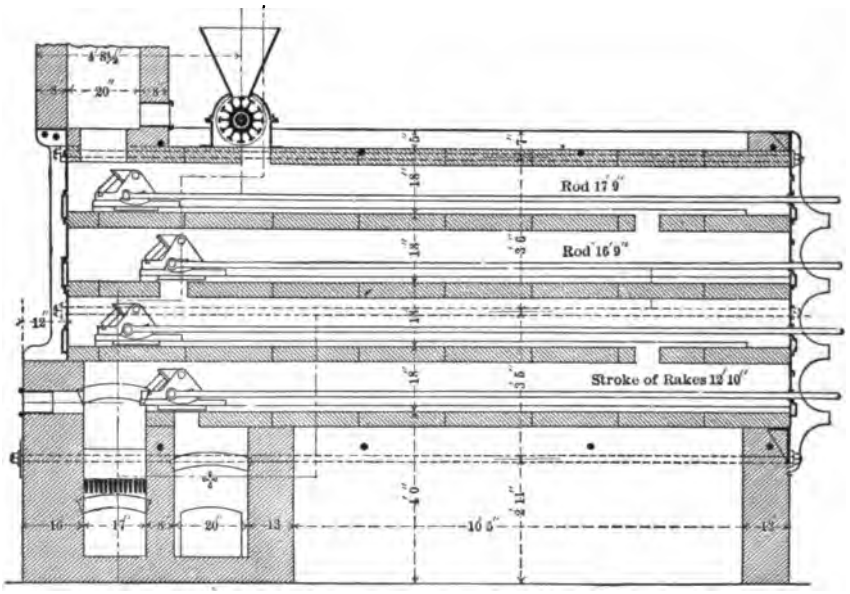


FIG. 116. HAMMOND-SPENCE FURNACE.
Longitudinal, vertical section.

attended to as many as six double furnaces. Transport of ore to and from the furnace is not included in either case.

In all cases the Spence furnace is built with a fireplace. When it is operated by the heat of the burning ore alone, carbonaceous fuel is used only to raise the furnace to the requisite temperature in starting a campaign. If extraneous heat is required throughout the operation a fire is maintained constantly on the grate and the furnace is then simply a mechanical reverberatory. It is converted into a muffle furnace by arranging combustion flues under the lowest hearth.

The furnace illustrated in Fig. 115 is the Hammond-Spence. A single furnace is 17 ft. 6 in. long and 9 ft. 6 in. wide outside, and requires a floor space 40×20 ft. and head room 18 ft. high. In setting the furnaces in a row a space of 5 ft. should be left between them. Four furnaces require a building 40×65 ft. Each furnace has a hydraulic cylinder for moving the rakes, which is operated by a pump located at any convenient place. About 5 h. p. are required as the maximum. In the construction of the furnace there are needed 16 T-rails (60 to 70 lb.) 16 ft. long, eight of 17 ft. 6 in., two of 12 ft., 25,000 lb. of cast and wrought iron parts, 40,000 lb. of special fire brick and tiling, 10,000 red brick, 10 bbl. cement, 10 bbl. lime, and the labor of three masons and two helpers for nine days, and one machinist and his helper for 12 days.

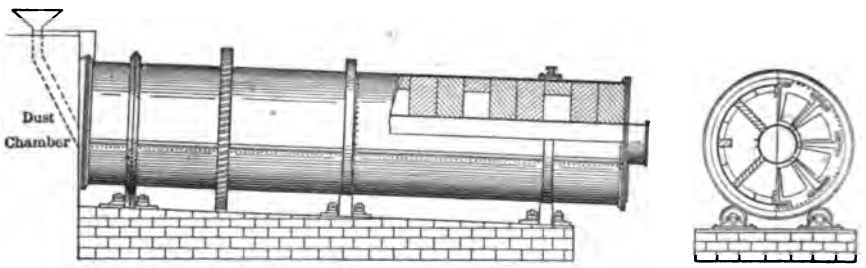
The furnace illustrated in Fig. 116, which is reproduced from Peters' *Modern Copper Smelting,* is also a Hammond-Spence, used in the Treadwell mill, Douglas Island, Alaska, for roasting auriferous pyrites concentrates as a preliminary to chlorination. According to Peters, six double furnaces roast 36,000 to 40,000 lb. of concentrates per day to a "dead roast," with the consumption of about ⅛ cord of wood per 2000 lb. of ore. Six double furnaces are attended by two men per shift, of whom one is occupied in keeping the ore hoppers full and the other regulates the temperature of the furnace. The iron rails which gave much trouble in the older Spence furnaces are in this replaced by very hard brick tiles.

Wetherill Furnace.—This is a recently patented muffle furnace, especially intended for blende roasting, which is designed on scientific principles, though it is very complicated in construction, and in its parts is reminiscent of other well known furnaces. The patent drawings show a long roasting chamber, in which the ore is moved forward by means of a continuously traveling rabble, the arms of which extend through slots in the sides, as in the Wethey furnace; the slots are closed by tripping gates in the same way. The roasting chamber, or muffle, is heated by flues above and below it, the gas which burns in the flues being supplied by producers arranged symmetrically along the sides of the furnace and the air for the secondary combustion being preheated by the waste gases in counter current recuperators built under the furnace. The roasting chamber is also supplied with heated air which is supplied at intervals in its length. The sulphurous gas is also led off at intervals. The arrangement of the furnace is such that the temperature can be closely controlled in its different parts, the roasting chamber being considered as consisting of an initial portion in which the ore is supplied with hot air and where the external combustion flues are maintained at a fairly high temperature, in order to raise the ore rapidly to a dull red

heat, an intermediate portion in which the desired temperature is afforded by the burning of the ore itself, and there is no necessity for so high a temperature in the external heating flues, and a final portion, wherein hot air is again admitted and the external combustion flues are raised to a high temperature in order to effect the removal of the last of the sulphur.[1]

Revolving Cylinder Muffle Furnaces.

Douglas Furnace.—The Douglas central flue, revolving cylinder furnace, which has given excellent results in desulphurizing copper ore in Arizona, has never been applied to blende roasting, but its obvious adaptability to that purpose and the fact that a similar furnace has been used experimentally in Upper Silesia make it worth while to present a brief description of it. The distinguishing feature of this furnace is a fixed central flue, which



FIGS. 117 AND 118. ELEVATION AND SECTION OF DOUGLAS 80-IN. FURNACE.

revolves with the furnace, and carries the gases, when fuel is used, from the fireplace to the chimney, heating by radiation the annular space in which the ore is roasted.

The flue may be made of tile or of cast iron, and is supported throughout its length by tiles or iron plates, to the number of four in the 60 in. diameter furnace and eight in the 80 in. diameter furnace. The section of the 60 in. furnace therefore exhibits four compartments in which the ore is roasted surrounding a fifth, which connects the fireplace with the smoke-stack; or in the 80 in. furnace, eight roasting compartments and a ninth which serves as a flue. In order to increase the agitation of the ore and expose it more effectually to the air, it is made to fall from compartment to compartment. To effect this the plates or tiles which support the central flue are perforated at intervals with slots, or an opening is left between alternate tiles.

[1] John Price Wetherill, United States patent No. 678,078, July 9, 1901.

The central flue may be constructed of cast iron pipe supported by spiders, and the ore may be agitated by shelves, as in the ordinary revolving cylinder furnace. The square or rectangular tile flue, supported by heavy tiles built into the lining, is the best storer and equalizer of heat, but cast iron may be used for both flue and supports. If the tiles be of good material and well locked together in the cylinder, the flue and its supporting shelves will not work loose or fall to pieces. The admission of air to the roasting ore is regulated by a register at the discharge end.

Experience has demonstrated that a boiler plate shell is preferable to cast iron. The wrought iron shell will not buckle if its rigidity be increased by inserting six or eight rails, or light beams, between the shell and the rings upon which the furnace revolves.

The rate of speed varies from one revolution in three minutes to one revolution in 10, depending upon the character of the ore under treatment.

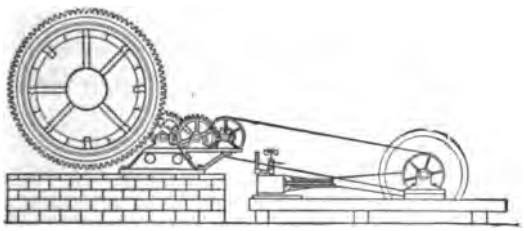


FIG. 119. DRIVING GEAR OF DOUGLAS FURNACE.

Owing to the frequent and thorough agitation of the ore, a great deal of dust is made as it falls from compartment to compartment, an objection to its use in the manufacture of sulphuric acid. A dust chamber is therefore a necessary adjunct. The 60 in. furnace, when roasting pyrites fines free from lead and with but a small percentage of copper, will reduce the sulphur to 2% and treat 20,000 lb. per day. Ores which are likely to sinter, such as galena or matte rich in lead, or even the higher grades of copper matte, cannot safely be roasted in the confined and somewhat inaccessible space of the roasting compartments of the cylinder.

The wear and tear, and therefore the cost of repairs is claimed to be less than in the rake furnaces, and the weight of the structure, and therefore the wear on the rings and rollers, is not as great as in the Brückner cylinder of equal capacity. If the furnace be well set up and kept well in line 2 h. p. will rotate a 60 in. cylinder, and 3 h. p. an 80 in. cylinder.

Koehler Furnace.—The Koehler revolving cylindrical muffle furnace[1] was

[1] Patented by Rudolph Koehler, Lipine, D. R. P., 57,522, May 7, 1890.

used experimentally at Lipine, Upper Silesia, in 1893. This furnace consists of a large brick-lined iron cylinder, supported on rollers in the ordinary manner, with closed ends against which abut closely separate sections (to be referred to subsequently), which do not revolve with the cylinder, but being mounted on wheels can be moved on a line parallel to the axis of the cylinder when it is necessary to have access to the interior of the latter. The cylinder is divided into three concentric sections. The innermost is the roasting chamber proper; surrounding it are the combustion flues, and outside of them are flues for preheating the air which is to enter the roasting chamber. Both the combustion flues and the air flues are so arranged that the gas and air make a return pass through them, traveling forward and back in the direction of the axis of the furnace; i.e., if the gas in the flue which is uppermos is going east, that in the lowermost flue, 180° from the former, is going west. This reversal in direction is effected in chambers in the stationary end sections of the furnace, which have ports corresponding to the flues in the cylinder. By a similar arrangement the air is delivered into the roasting chamber. The air and flames travel always in opposite directions. The cylinder is arranged with mechanical feeding and discharging devices.

This furnace was said to give satisfactory results as to capacity and degree of desulphurization, but so far as I know the original furnace at Lipine has not been duplicated, probably because of the complicated character of the design and the high cost of installation. It is interesting, however, as furnishing a precedent for the use of revolving cylinders in blende roasting and indicating the adaptability of the excellent furnace designed by James Douglas, which has been described previously. The Douglas furnace is obviously far more simple and more rational in its construction than the Koehler furnace, in which the inventor apparently tried to do too much in the direction of fuel economy.

V.

UTILIZATION OF THE SULPHUROUS GASES.

The sulphurous gases developed in blende roasting are injurious both to human health and to vegetation, and in certain countries, especially Germany and Great Britain, their neutralization is required by law. In the United States they may still be discharged into the atmosphere, but the self interest of the proprietors of the works compels this to be done through the medium of moderately high chimneys, whereby the gas is so diluted before descent to the earth that it is no great nuisance. This method of disposing of the waste gases has not been carried by any means so far in the United States, however, as in Germany, where there is a chimney 142 m. high at Freiberg in Saxony, one 100 m. high at Hamborn, and many nearly as high in Upper Silesia. Since the increase in the draught efficiency of a chimney of greater height than 40 m. is insignificant, the extra heights of the chimneys mentioned above are to be regarded simply as means of dispersing the noxious gases. Various plans for the neutralization of these gases by chemical agents, such as absorption in water, in chamber sulphuric acid, in a solution of calcium polysulphide, and by contact with zinc oxide, milk of lime and several other substances have been tried, but none of these except the milk of lime method has been successful in practice. For an account of experiments in this direction the reader is referred to Schnabel's *Handbuch der Metallhüttenkunde,* Vol. II, p. 58 et seq.; Schnabel, *Preuss. Zts. f. Berg- Hütten.- u. Salinenwesen,* 1881, XXIX, p. 395; Winkler, *Freiberger Jahrbuch,* 1880, p. 50; Hasenclever, *Fischer's Jahresbericht,* 1881, p. 173, and 1886, p. 257; *Zts. des Vereins deutscher Ingenieure,* 1886; and C. A. Hering, *Die Verdichtung des Hüttenrauches,* 1888. This problem, it should be understood, concerns only the disposition of the gases of reverberatory furnaces which are low in sulphurous anhydride, seldom containing as much as 2% by volume, and are contaminated, moreover, by the combustion gases of the fuel and particles of soot. The pure and rich gas of muffle furnaces is easily condensed by conversion into liquid sulphurous acid or sulphuric acid, both of which are valuable commodities; wherefore

it follows that zinc producers in using reverberatory furnaces are not only creating a nuisance, but also are wasting a valuable part of their ore, if their works be situated in proximity to a market for sulphuric acid or products which can be made from it. In Belgium and Germany the recovery of sulphuric acid from blende roasting is now practised so generally that there is an overproduction and some difficulty in disposing of the product. In the United States sulphuric acid is produced by only four zinc smelters, two of whom but recently engaged in the business.

Neutralization with Milk of Lime.

Both sulphurous and sulphuric anhydrides, SO_2 and SO_3, combine with milk of lime (calcium hydrate, CaO, H_2O), forming calcium sulphite, $CaSO_3$, and calcium sulphate, $CaSO_4$, respectively. The sulphite on exposure to the air changes gradually into sulphate. The latter is valuable as a fertilizer and for the manufacture of plaster of paris or stucco, but since natural gypsum is abundant and is seldom worth more than \$1@\$1·25 at the quarries, while stucco is worth only \$3@\$5 per 2,000 lb. f. o. b. works in the United States, the market for the artificial product is limited. Consequently the neutralization of sulphurous acid by means of milk of lime cannot be regarded as a method of recovering a valuable by-product, but must be considered merely a means of overcoming a nuisance. At the Hohenlohehütte and Kunigundehütte, in Upper Silesia, where this process is employed extensively, the accumulation of calcium sulphate is far in excess of what can be marketed, and causes a large outlay for its storage besides the expense of its production. As a method of neutralizing sulphurous fumes, however, the employment of milk of lime is highly efficient and is adaptable to the low grade and carbonaceous gases of reverberatory furnaces.

In carrying out this process an excess of milk of lime is required. At the Hohenlohehütte it is made to descend in a spray through a series of towers, each 6 m. high, the sulphurous fumes being drawn up through this spray by the draught of a chimney 100 m. high, which disperses the gases escaping uncondensed from the towers. The precipitate and liquor drawn off from the bottom of the towers are conveyed to sumps, where the former is allowed to settle, after which the excess of milk of lime is pumped again to the towers for further use. According to Doctor Grosser, of Kattowitz, the precipitate has the following composition: CaO, 37·75%; MgO, 1·45%; Al_2O_3, 4·14%; Fe_2O_3, 1·10%; SO_2, 38·4%; SO_3, 2·85%; CO_2, 4·15%; SiO_2, 5·53%; H_2O, 3·40%; total, 98·77%. This analysis indicates the presence of 72% of calcium sulphite and 4·84% of sulphate. According to Kosmann, the general

average is 34% SO_2 and 5% SO_3, corresponding to 64% $CaSO_3$ and 8·5% $CaSO_4$.[1] By this method 75 to 80% of the sulphurous anhydride is converted into calcium compounds, the remainder escaping through the chimney.

The sulphurous gases from reverberatory furnaces may also be neutralized by passage through towers packed with broken limestone, or dolomite, down which water is trickling, calcium (or magnesium) sulphite being formed as with milk of lime; but a large cross-section of tower, strong draught and great quantity of water are required, and the method is far less efficient than the milk of lime method and is nowhere in use among zinc works.

Sulphurous anhydride is soluble in water, H_2SO_3 being formed, and a weak solution can be made in a simple coke or quartz filled tower, but with the comparatively weak gases from reverberatory furnaces the absorption is incomplete, even with preliminary cooling of the gas and the use of towers of large cross-section.[2] Of all methods that have been tried for the neutralization of sulphurous gas from reverberatory furnaces the use of milk of lime is certainly the best, and even it is not profitable commercially, being valuable only as a means of getting rid of a harmful product.

At Flône, in Belgium, the sulphurous gases from blende roasting furnaces used to be passed into waste heaps of alum shale, whereby the alumina of the latter was rendered soluble as sulphate. The employment of such a process is obviously possible only under exceptional local conditions. The utilization of the sulphurous gas for the manufacture of alum is still carried out in a unique manner at the blende roasting plant of L. de Laminne, at Ampsin, Belgium. The gases discharged by the eight 3-hearth reverberatory furnaces which are comprised in that plant pass into a network of flues built on the surface out of alumiferous schist, which was dug and once treated for the extraction of alum at the end of the eighteenth and beginning of the nineteenth centuries. This schist, which originally contained pyrite, was at that time roasted in heaps for the production of aluminum sulphate, which was then leached out with water. The residues from those old operations contain no more sulphur, but are still able to absorb readily sulphurous acid when exposed to it. For this purpose absorption galleries, several tiers high, are built of it, and upon passage of the gas through them the walls become impregnated with aluminum sulphate, which can be subsequently extracted by lixiviation. There are arranged two main galleries, from which branch out the absorption galleries in the form of cul-de-sacs. At the end

[1] Oberschlesien, sein Land und seine Industrie, p. 199.

[2] The capacity of water to absorb sulphurous anhydride gas diminishes with the strength of the gas. With pure gas, water at 15° C. will take up about 15 kg. per cu. m. (15%). With gas containing 1% SO_2 it will take up only about 1·5 kg. per cu. m.

of the main galleries there is a chimney 50 m. high, of which the foot is about 50 m. higher than the floor of the roasting furnace house. This chimney is not used to produce a constant draught on the roasting furnaces, but only to regulate the latter in case of need.[1]

Recovery of Sulphur as Liquid Sulphurous Anhydride.

This process is based on the power of cold water to absorb or dissolve cool sulphurous anhydride gas and the ability to disengage the gas thus dissolved by heating and agitating the solution. This principle is applicable to the ordinary roast gases from reverberatory furnaces, since substantially none of their constituents besides the sulphurous anhydride is soluble in water, and consequently a separation is possible, but practically the process cannot be worked to advantage with gases of a lower tenor in SO_2 than 4% by volume, and it is therefore limited to the uncontaminated gases from muffle furnaces. The invention of a commercially successful process for carrying out this idea is due to Messrs. Haenisch & Schroeder, by whom it was patented in 1883,[2] since which time it has come into use at several zinc works in Westphalia and Upper Silesia, particularly at the Grillo works at Hamborn, Westphalia, and at the Silesia works at Lipine and the Guidotto works at Chropaczow in Upper Silesia, although at the last it has been lately discontinued in favor of sulphuric acid manufacture. In the United States it is in use only at the works of the Somerset Chemical Co., at Bound Brook, N. J., where the sulphurous gases are obtained by burning brimstone.

Haenisch & Schroeder Process.—This consists in the absorption of the sulphurous anhydride of the roast gases in water dripping down through a tower, expulsion of concentrated anhydride mixed with only a little aqueous vapor from the solution thus obtained, separation of the aqueous vapor by condensation and absorption in sulphuric acid or calcium chloride, and liquefaction of the pure sulphurous anhydride.[3]

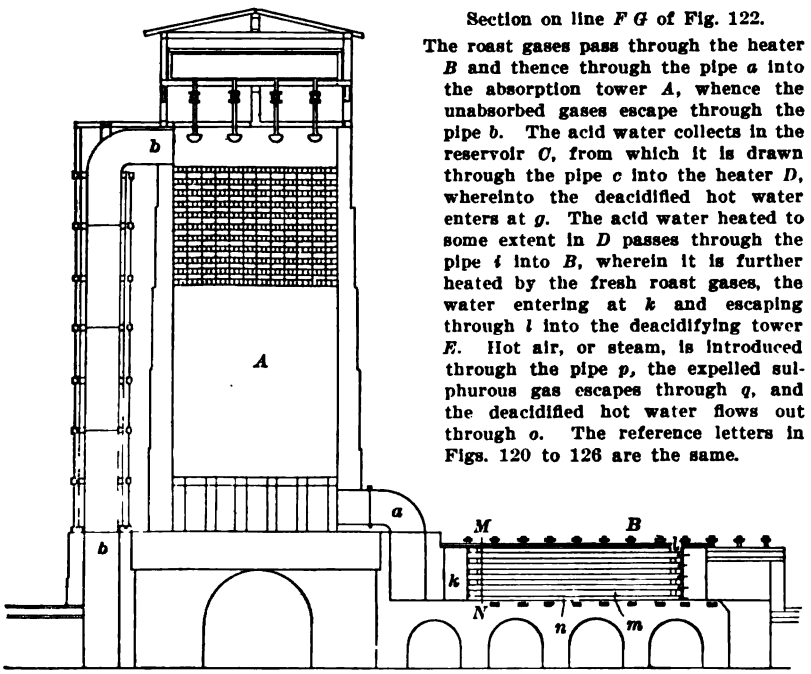
Absorption Tower.—The gases from the roasting furnace, which will probably have a temperature of about 400° C., are conducted to the absorption tower passing on the way through a flue designed to cool them to a certain extent for the triple purpose of utilizing their heat, reducing their

[1] Ad. Firket, Annales des Mines de Belgique, VI, ii, 235 and 236.

[2] German patents, 26,181, 27,581 and 36,721.

[3] For more specific details as to this process reference may be made to Zts. f. angew. Chemie, 1888, p. 448; Lunge on the Soda Industrie, I, 264; and a paper by K. Eilers in Trans. Am. Inst. Min. Engineers, XX, 336; also to a paper by Dr. Steger in Zts. f. d. Berg- Hütten.- u. Salinenwesen im Preuss. Staate, I., iii, 506, and a monograph entitled "Flussiges Schwefeldioxid" by August Harpf in Sammlung chemischer und chemisch-technischer Vortrilge, 1900, V, 235 to 414. The last is the most complete treatise in existence on the properties, recovery and uses of sulphurous anhydride.

volume and lowering their temperature so as to make more easy their absorption and retention by the water in the tower. The tower, which may well be as high as 35 m., was filled originally with coke, but in later designs it is filled with coke only in the upper half, the lower being fitted with a series of earthenware plates, as shown in the accompanying engraving.[1] The slightly cooled gases enter the tower at the bottom and rise through the rain of water, to which they give up their sulphurous anhydride, while the unab-



Section on line *F G* of Fig. 122.

The roast gases pass through the heater *B* and thence through the pipe *a* into the absorption tower *A*, whence the unabsorbed gases escape through the pipe *b*. The acid water collects in the reservoir *C*, from which it is drawn through the pipe *c* into the heater *D*, whereinto the deacidified hot water enters at *g*. The acid water heated to some extent in *D* passes through the pipe *i* into *B*, wherein it is further heated by the fresh roast gases, the water entering at *k* and escaping through *l* into the deacidifying tower *E*. Hot air, or steam, is introduced through the pipe *p*, the expelled sulphurous gas escapes through *q*, and the deacidified hot water flows out through *o*. The reference letters in Figs. 120 to 126 are the same.

FIGS. 120 AND 121. ABSORPTION TOWER AND HEATER.

sorbed nitrogen and oxygen escape through a flue in the side of the tower near the top. The gases which enter the towers at Hamborn and Lipine contain 6% SO_2 by volume; the escaping gases contain only 0·05%; the water drawn off from the towers contains 12 kg. of SO_2 per cubic meter, i.e., 1·2% by weight.

Expulsion of the Gas from the Aqueous Solution.—In order to drive off the gas from the aqueous solution it is necessary to raise the latter to a temperature of 100° C. To accomplish this it is made to pass first through

[1] German patents, 36,721 and 52,025. Vide Fig. 127.

a lead box divided by a series of horizontal diaphragms of sheet lead, supported by intermediate bars of lead laid parallel with the long axis of the box, which is thus subdivided into a series of longitudinal cells. Water at a temperature of nearly 100° from which the acid has just been expelled in a subsequent part of the system is run through the second, fourth, sixth, eighth, tenth, twelfth and fourteenth horizontal sets of cells, while the acid water from the tower is passed in the reverse direction through the alternate rows, its temperature being raised thereby to 70° or 80° C. It is then con-

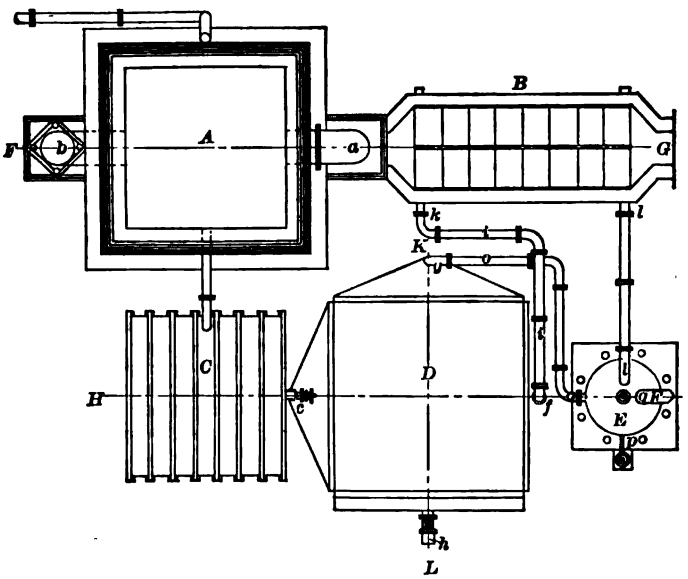


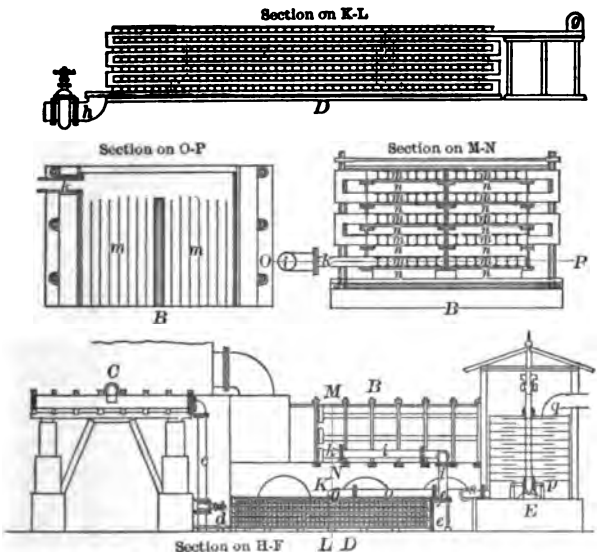
FIG. 122. DIAGRAM OF ARRANGEMENT OF SULPHUROUS ANHYDRIDE APPARATUS.

veyed to a similar box, except that the cells are of larger cross-section and the horizontal lead plates are supported by sheets of iron, or else to a simple covered lead pan, placed on the main flue from the roasting furnaces, wherein its temperature is raised to 100° C. and its contents of sulphurous acid are given off.[1]

Liquefaction of the Gas.—The sulphurous gas thus expelled, with which some aqueous vapor is admixed, is led through a pipe to a water cooled

[1] The capacity of water to hold sulphur dioxide decreases with increase of temperature. At 20° C. it will hold 8·6% SO_2; at 30° C., 7·4%; at 40° C., 6·1%; at 50° C., 4·9%; at 60° C., 3·7%; at 70° C., 2·6%; at 80° C., 1·7%; at 90° C., 0·9%; and at 100° C., 0·1% (Harpf, op. cit.)

worm, in which part of the steam is condensed and drained off, whence the gas passes on to a small tower packed with coke saturated with concentrated sulphuric acid, in which the last part of the water is removed. The dry gas is then compressed into a liquid by a suitable pump (made of bronze) under a pressure of 2 to 3·5 atmospheres, the liquefaction taking place in a suitably designed water cooled worm, which absorbs the heat of compression. For regulation of the compression a taffeta sack is interposed in the pipe from the drying tower to the pump, and the movement of the latter is regulated according to the size of this sack. The liquid anhydride from



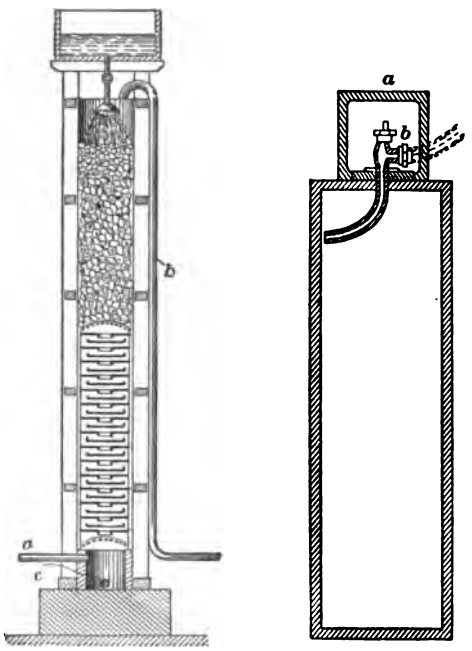
FIGS. 123 TO 126. SECTIONS OF SULPHUROUS ANHYDRIDE PLANT.

the worm runs down into a wrought iron reservoir, whence the nitrogen and oxygen, which are not liquefiable at the temperature and pressure at which sulphurous anhydride is liquefied, escape through a properly arranged valve into a pipe leading to the main absorption tower. The liquid anhydride, 99·8% pure, is tapped from the reservoir into steel cylinders containing 50 or 100 kg. At the Guidottohütte, in 1891, about 90% of the sulphur content of the gas was recovered.[1]

Treatment of the Residual Water.—The residual water in the lead pans where the acid solution is boiled still contains a small percentage of sul-

[1] Th. Dahlblom, Berg -u. Hüttenm. Ztg., Dec. 11, 1891, p. 449.

phurous anhydride. In order to recover the latter the water is conducted to a column, down which it is made to fall in a spray against a current of steam, which is introduced at the bottom of the column. The gas which is thus expelled is conducted from the top of the column to the main stream from the boiling pan, while the hot water drawn off from the bottom of the column is conducted to the lead cells for heating the solution from the absorption tower, as previously described. The tower shown in Fig. 126 has a vertical shaft, carrying horizontal discs which alternate with rings fixed in the tower. The revolution of the shaft sprays the water by centrifugal force.



FIGS. 127 AND 128. SULPHUROUS ANHYDRIDE APPARATUS.

Fig. 127: Absorption tower with earthenware plates in lower half, coke filled in upper half.
Fig. 128: Cylinder for holding liquid anhydride.

Receiving Cylinders.—The cylinders for receipt of the liquid anhydride are made of wrought iron or steel, tested to withstand a pressure of 735 lb. per sq. inch (50 atmospheres), in the form shown in the accompanying engraving. The stop cock at the top is protected by a removable cap. In drawing off the anhydride, when it is desired in gaseous form, the cylinder is stood on end and the stop cock opened, the gas flowing until the tempera-

ture inside the cylinder has fallen to —10° C., when the flow ceases until the temperature has risen again; to obtain the anhydride in liquid form the cylinder is laid on its side with the spout of the stop cock pointing upward, in which manner nearly the whole content of the cylinder can be drawn off as liquid. The tension of the liquid anhydride is 0 at —10° C., 1·26 atmospheres at +10° C., 2·24 at 20°, 3·51 at 30° and 5·15 at 40°. It is best therefore not to use it in places where the temperature is higher than 40° C., which is the ordinary heat of the sun in summer in the temperate zone.

Commercial Conditions.—According to Kosmann, the sulphurous acid plant at Lipine, Upper Silesia, which was built in 1887, cost about 250,000 marks ($62,500), its capacity being about 1250 metric tons per annum. Its production in 1900 was 1306 metric tons, valued at 40 marks per ton (equivalent to $8·63 per 2000 lb.).

Liquid sulphurous anhydride is consumed chiefly in the manufacture of sulphite wood-pulp, in the manufacture of beet sugar, glue and gelatine, and for general bleaching and disinfecting purposes. The demand from those sources is still rather limited, and notwithstanding the technical excellence of the Haenisch & Schroeder process it does not appear to be a great commercial success. As previously remarked, it has been abandoned at the Guidottohütte, in Upper Silesia, although it continues in use at the Silesiahütte. The latter made 1090 metric tons in 1901, valued at 43,602 marks ($10,377 = $9·52 per 1000 kg.). The proceeds scarcely covered the cost of production.[1]

Porák Process.—A. Harpf, of Przibram, Bohemia, in a paper on the utilization of waste gases of metallurgical works, read before the International Chemical Congress at Vienna in 1898, described a method of utilizing sulphurous anhydride gas, which is employed by Porák Brothers, of Kienberg, Bohemia. The gas is enriched, if necessary, by burning sulphur in it. It is then cooled and conveyed through lead pipes to a water injector which forces the gas down to a cylinder 20 ft. below. A centrifugal pump raises the liquid again to the injector, and this circulation is continued until the pressure of the gas and air above the water in the cylinder, which can absorb only a certain proportion of SO_2, forces the gas from the tank over into a system of vats filled with milk of lime and provided with stirrers. As the lime dissolves as calcium bisulphite, the liquid becomes clearer; the solution of calcium bisulphite is finally drawn off ready for use in the preparation of wood pulp for paper manufacture.[2]

[1] Statistik der Oberschl. Berg-u. Hüttenwerke, 1901, p. 86.

[2] This process is also described in the paper in Sammlung chemischer und chemisch-technischer Vorträge previously referred to.

Recovery of Sulphur as Sulphuric Anhydride.

The sulphurous gases given off in burning galena and pyrites ores at the Muldnerhütte at Freiberg and the Rhenania chemical works at Stolberg in Germany have been for a long time utilized for the preparation of sulphuric anhydride and the gases from blende roasting are equally applicable for that purpose.[1] In this process the sulphurous gases are first passed through towers packed with coke or quartz through which sulphuric acid is trickling in order to remove aqueous vapor, after which they are conveyed to externally heated vessels containing platinized asbestos, in which the sulphurous anhydride and oxygen combine to form sulphuric anhydride by catalytic action. The gases containing the latter are led through towers in which it is absorbed by concentrated sulphuric acid. Pure sulphuric anhydride is then obtained by distillation of the solution. The sulphurous anhydride which escapes absorption in the towers is conducted to lead chambers for the manufacture of sulphuric acid. This process is chiefly interesting because of the recent improvements in its execution whereby the direct manufacture of sulphuric anhydride from sulphurous by catalytic action and the preparation of monohydrate of any desired strength has become a process of commercial practicability, which promises to revolutionize the sulphuric acid industry.

Manufacture of Sulphuric Acid.

The most practical and the most profitable method of utilizing the sulphurous gases from muffle roasting furnaces is the manufacture of sulphuric acid, provided the works be situated in a region where there is a market for that product; (1) because the extra cost of production is small, and (2) the product is one which is in demand in many branches of industry and can be marketed easily at a fair price when the producer is favorably situated. An unfavorable geographical location of a zinc smelting works is in fact the only obstacle to the recovery of sulphuric acid as a by-product. In Upper Silesia where acid is made by several producers of spelter, the freight rates on acid are so high that markets at a distance greater than 100 miles cannot be reached,[2] and the outlet for the product being thus limited, its production is necessarily restricted, which also restricts the mining of blende, since the government grants permits for the building of roasting plants only when the gases are used in the manufacture of sulphuric acid or otherwise neutralized. The recovery of sulphuric acid by the smelters in Kansas and

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 81.

[2] United States Geological Survey, 19th annual report, part VI, p. 238.

Missouri has not yet been generally undertaken,[1] partly because of remoteness from large markets and partly because of the large cost required to install an acid plant. On the other hand, sulphuric acid has been made for many years by the large smelters of Illinois, who are more favorably situated geographically.

THE MARKET FOR SULPHURIC ACID.—The manufacture of sulphuric acid is one of the most important branches of the world's industry. It is more than an important branch of the chemical industry alone, because it is the basis of the entire chemical industry. Without it, it would be impossible to manufacture a multitude of chemical products which are of common use in the arts. Considering the two elements which exist in zinc blende, namely, zinc and sulphur, there is no question that in the requirements of the world's commerce the sulphur is the more important; that is to say, if it were necessary to dispense with either of them zinc could be spared better than sulphur. The value of the annual product of sulphuric acid in the United States is considerably greater than that of spelter.

Production in the United States.—The production of sulphuric acid in the United States in 1900, as reported by Messrs. Charles E. Munroe and Thomas M. Chatard in Bulletin No. 210 of the Twelfth Census, was as follows:

Grade.	Short tons.	Value.	Per ton.
50° B.	953,439	$7,765,852	$8·35
60° B.	17,011	246,284	14·47
66° B.	377,279	6,035,069	16·00
Total.	1,347,729	$14,247,185	$10·57

The above statement does not represent the actual make of acid of the various grades. The figures of quantity and value of the 50° acid include that which was made at certain works and consumed there in the manufacture of fertilizers. This was really acid of 50° strength. Its amount was 789,359 tons. In addition thereto a large quantity of acid of different grades consumed at works for various purposes has been included with the actual 50°, of course reduced to that grade. It is to be inferred that the quantities of 60° and 66° acid reported by the Census include only what was marketed as such. The values are determined on the basis of the output at the works. If the entire product of acid were reduced to 50° B.

[1] A plant erected at Iola, Kan., by the Standard Acid Co., was put in operation in 1902 and is so far the only zinc-acid works west of the Mississippi River. This company subsequently consolidated with the Southwestern Chemical Co., of Kansas City, under the title of United Zinc and Chemical Co.

the quantity would have been 1,540,623 tons, equivalent to 1,232,500 tons of 60° B., or 1,027,080 tons of 66° B., or 955,184 tons of pure H_2SO_4. This does not include the acid made from blende and as a by-product in the treatment of gold and silver bearing pyrites at certain metallurgical works in the West, the quantity of which in 1900 was reported by *The Mineral Industry* as 85,000 tons, basis 66° B., equivalent to 127,500 tons of 50° B. According to the Census report on the smelting industry, the acid product of zinc works in 1899 was 58,827 tons (grade of acid not stated, but probably of 50° and 60° B.).

Purposes for Which Used.—The largest single channel of consumption of sulphuric acid is in the manufacture of acid phosphate of lime from phosphate rock to serve as a fertilizer. The next important use is in the refining of crude petroleum. Other large uses are the manufacture of alum, aluminum sulphate, ammonium sulphate, sodium sulphate and chlorhydric acid, nitric acid, potassium sulphate, potassium bichromate, nitroglycerin and glucose. The preparation of iron and steel for galvanizing and tin plate manufacture requires a large quantity. Other uses are the manufacture of carbonic acid, copper sulphate, zinc vitriol, copperas, parting gold and silver, and the manufacture of magnesium sulphate, besides a multitude of less important uses.

Sulphuric acid is used in the arts in four grades of strength, namely, acid of 50° B, which contains 62·177% H_2SO_4; acid of 60° B, which contains 77·605%; acid of 66° B, which contains 93·5%; and acid of 96 to 97%, which is required in the manufacture of nitroglycerin. The petroleum refiners need an acid of 66° B strength. In almost all other branches of the chemical industry an acid of 60° B strength is sufficient and in many 50° B is strong enough. Of course, in cases where 50° B acid can be used, acid of 60° B is also applicable by dilution. For most purposes acid must be free from arsenic and selenium, but that is not a requisite in fertilizer manufacture. The manufacturers of fertilizing material use generally acid of 50° B strength, and in order to avoid the expense of paying freight on the large quantity of water which that acid contains, fertilizing works are generally established in close proximity to sulphuric acid works or a supply of the raw material for acid making. The manufacturers of acid phosphate of lime and superphosphate are situated chiefly on the Atlantic Coast and in the Southern States, and to a large extent make their own acid and consequently are out of the general market. There are, however, a good many small phosphate works scattered about the country which have to buy their acid. In petroleum refining a large part of the sulphuric acid used is recovered in a foul and dilute form commonly known as sludge

acid, which can be reconcentrated or used for the manufacture of fertilizing material; such recovery is practiced to a considerable extent. The consumption of phosphate rock in the United States in 1900 was 815,855 short tons, the decomposition of which required 815,855 tons of 50° B. acid.

Production of Sulphuric Acid in Germany.—According to the statistics of Robert Hasenclever the production of sulphuric acid, reduced to the basis of 60° B., in Germany in 1897 was 845,582 metric tons, of which 136,868 were derived from blende roasting. Statistics of the production of acid by Silesian zinc smelters in 1900, as reported by the Oberschlesischen Berg- und Hüttenmännischen Verein, are summarized in the following table, in which the figures of consumption and production represent metric tons:

Name of Works.	No. of roasting furnaces.	Lead Chambers.		Raw blende roasted.	Roasted blende product.	Niter used.	Nitric Acid Used	Sulphuric Acid Produced *c*		
		No.	Cu. m. Capacity					50° B.	60° B.	66° B.
Bernhardihütte	2	1	7,500	3,329	2,523	....	41	2,249	.....	.....
Guidottohütte	32	3	21,300	15,798	16,850	224	278	.....	14,009	.....
Laryhütte	24	4	7,200	15,141	11,752	207	286	1,113	7,162	.....
Reckehütte	*a* 28	8	48,242	55,060	44,022	469	...	4,189	7,363	10,005
Silesiahütte	16	5	18,500	32,293	25,834	*b*247	...	686	7,888	2,769
Total	102	21	102,742	121,621	100,981	1,147	605	8,237	36,422	12,774

a Besides 117 kilns. *b* Includes nitric acid. *c* All the works were in operation 52 weeks, except the Silesia, which ran only 48.

The average value of the acid produced was 12·1 marks ($2·88) for 50° B., 22·2 marks ($5·28) for 60° B., and 30·9 marks ($7·35) for 66° B.

Sulphuric Acid Making by the Chamber Process.—The manufacture of sulphuric acid is in itself too large a subject to be entered into from a technical standpoint in a treatise on the metallurgy of zinc, and with respect to it reference should be made to the classical work of Lunge on *Sulphuric Acid and Alkali,* and his subsequent papers in various volumes of *The Mineral Industry;* also to the paper entitled "The Manufacture of Chamber Sulphuric Acid," by F. J. Falding in *The Mineral Industry,* Vol. VII, and a paper by the same author on "The Manufacture of Sulphuric Acid Stronger than Chamber Acid" in *The Mineral Industry,* Vol. VIII. What concerns the zinc metallurgist in taking up this subject, is the economy that may be looked for in the recovery of sulphuric acid from the gases of his roasting furnaces. For an intelligent understanding of the subject, however, it is necessary to refer to the fundamental reactions which are involved in sulphuric acid manufacture.

The sulphur of the raw material, brimstone, pyrites or blende, must be converted first into sulphurous anhydride, SO_2, and the sulphurous anhy-

dride must be in some way given another atom of oxygen in order to produce sulphuric anhydride, SO_3, which will combine directly with water, forming sulphuric acid, H_2SO_4. Sulphuric acid was made originally by calcining copperas (ferrous sulphate), commonly called green vitriol, whereby sulphuric anhydride was given off, which was then absorbed in water, whence the name "oil of vitriol" by which sulphuric acid is often known. In burning brimstone, pyrites or blende, although there is reason to believe that sulphuric anhydride is formed in the furnace, if that be the case, it is subsequently reduced mostly to sulphurous anhydride, and the latter gas is what the acid maker has to deal with.

Chemical Reactions Involved.—The reoxidation of the sulphurous anhydride is accomplished by introducing it into large chambers of sheet lead supported by timber framework, together with steam and compounds of nitrogen and oxygen, the nitrogen-oxygen compound which plays the chief part in the transformation being the trioxide, N_2O_3. The change is, however, not one of simple direct oxidation, but involves a number of complicated reactions, of which the chief are expressed as follows:

I. Nitrate of sodium heated with sulphuric acid produces nitric acid and sodium hydrogen sulphate:

$$NaNO_3+H_2SO_4=HNO_3+NaHSO_4.$$

II. Nitric acid and sulphurous anhydride and water produce sulphuric acid and nitrogen trioxide:

$$2HNO_2+2SO_2+H_2O=2H_2SO_4+N_2O_3.$$

III. The sulphuric acid produced in a dilute form by the next preceding reaction is a final product; or it may be sent to the Glover tower for denitration of nitrosyl-sulphuric acid and concentration of itself; the nitrogen trioxide is absorbed in the Gay-Lussac tower in strong sulphuric acid, forming nitrosyl-sulphuric acid, (SO_2,NO_2,OH), or $SO_2\{^{NO_2}_{OH}$ thus:

$$N_2O_3+H_2O+2SO_3=2(SO_2,NO_2,OH).$$

IV. In the Glover tower the nitrosyl-sulphuric acid is decomposed by dilution with water, which is got from weak chamber acid:

$$2(SO_2,NO_2,OH)+H_2O=2H_2SO_4+N_2O_3.$$

V. The nitrogen trioxide formed by reaction IV continues around the circuit of reactions III and IV, the comparatively small quantity which is lost in the operation being replaced by reactions I and II.

Sulphuric acid made directly in the lead chambers is comparatively dilute,

being only about 50° to 52° B. in strength, and if there were no adjuncts to the chamber system the consumption of nitrate of soda would be excessive. A modern sulphuric acid plant, to produce an acid of higher tenor in H_2SO_4 with the minimum consumption of niter, comprises a system of towers known as the Glover and Gay-Lussac from the names of their inventors; there may be also so-called Lunge columns interposed between the series of chambers, which enable a reduction to be made in the size of chambers, and a fan to promote and regulate the draught.

Function of the Gay-Lussac Tower.—The Gay-Lussac is the last member of the sulphuric acid system. Its function is to absorb the nitrogen oxides, principally N_2O_3, which would otherwise escape from the last chamber and would be lost. This absorption is effected by causing the gas to rise through a tower packed with quartz, through which is falling a steady stream of cool sulphuric acid of about 60° to 62° B. The acid on leaving the tower should contain about 2·5% of N_2O_3, the quantity of acid falling through the tower being regulated so as to produce a liquor of that strength. In practice there are two or more Gay-Lussac towers to a set of chambers, the gas passing first into the bottom of the first tower, from the top of which it is led to the bottom of the second, and so on, until it leaves the last tower at the top, where it should be discharged preferably by means of a fan directly into the air. When a fan is used the system will be unaffected by outside meteorological conditions.

Functions of the Glover Tower.—The Glover tower, which is the natural corollary of the Gay-Lussac, has six functions, which are summarized by Mr. Falding in *The Mineral Industry*, VII, 651 to 702, as follows:

(1) "The denitration of the nitrous vitriol. The nitrous acid gas absorbed in sulphuric acid of 60° B. to 62° B. (141° Tw. to 150° Tw.) and recovered thus by means of the Gay-Lussac tower from the chamber process, is liberated and again put into circulation or made available for the chamber process by means of dilution and the action of the hot sulphurous acid gas in the Glover tower.

(2) "To supply the chamber process with nitrous acid gas lost during that process. When such loss is supplied by 'potting' nitrate of soda in the hot gas just before the entrance of the gas into the Glover tower, the Glover tower converts the nitric acid gas into nitrous acid gas and secures its thorough admixture with the sulphurous acid gas. When, however, such loss of niter is made good by the addition of nitric acid, nitrous vitriol, or waste nitrous acid of any kind to the nitrous vitriol from the Gay-Lussac supplied to the Glover, such acids are acted upon in the same way as the regular supply of nitrous vitriol, and this is by far the most efficient, direct

and most easily controlled method of supplying the chamber process with nitrous acid gas.

(3) "Concentration. The nitrous vitriol to be properly denitrated must be diluted. The weak acid or water thus added is evaporated by the heat of the hot burner gas and converted into steam which, passing into the chambers, supplies a part of the steam necessary for that process, and the acid thus concentrated to 60° B., or 62° B., is again ready when cooled for reabsorbing nitrous acid gas in the Gay-Lussac tower, while the excess, which is by far the major part, is ready for market or for concentration to acid of 66° B. in the concentrating department of the works.

(4) "Cooling of the furnace gas or the absorption of the heat of combustion in the furnaces. This heat if allowed to pass into the chamber system would, when added to the heat of oxidation of the SO_2 to SO_3, be detrimental not only to the process, but also to the leaden chambers.

(5) "Purification of the gas from impurities mechanically carried over from the furnaces. A considerable deposition of arsenic and selenium takes place in the Glover tower and except in extreme cases nearly all the dust carried over from the settling chambers is also retained, being washed out of the tower with the acid either in solution as sulphates or in suspension.

(6) "The oxidation of SO_2 to SO_3 in the highly nitrous atmosphere of the upper part of the tower, thus preparing the gas for rapid formation into sulphuric acid in the lead chambers."

Thus the nitrous acid gas absorbed by the Gay-Lussac tower and recovered from the chamber process is made available again by the Glover tower for use in the chamber process and the denitrated acid is concentrated for use in the Gay-Lussac for again absorbing nitrous acid gas. Incidentally, the sulphurous acid gas from the roasting furnaces is cooled and purified. Consequently the Glover tower is a means of heat recuperation; it owes its efficiency to the waste heat of the combustion of the ore which it recovers. For its successful operation, therefore, the sulphurous acid gas must not be permitted to cool below a certain temperature. The temperature of the gas entering the tower where no unusual precautions have been taken to conserve the heat, is from 316° C. (600° F.) to 427° C. (800° F.). The temperature of the gas leaving the tower is from 88° C. (190° F.) to 93° C. (200° F.). Since the heat produced by the oxidation of the sulphurous anhydride is about 93° C. (200° F.) to 99°C. (210° F.) in the zone of greatest activity, it is evident that the heat of the sulphurous gas coming from the roasting furnaces is nearly all if not entirely absorbed by the Glover tower, and the heat in the escape pipe of the Glover is due chiefly to the oxidation of sulphurous anhydride.

Degree of Concentration Effected in the Glover Tower.—According to Mr. Falding (loc. cit.) the heat produced by the oxidation of most ores used for making sulphuric acid is sufficient if properly conserved to concentrate the acid produced to 66° B., or 93·5% H_2SO_4, the difficulties in the way of such high concentration being entirely mechanical and as follows: (1) The conservation of the heat in the flues leading to the Glover tower; (2) the proper preparation of the Glover tower and its suitable construction, involving the use of sufficiently refractory material; (3) the purification of the concentrated acid; (4) the reconciliation of the functions of high concentration and proper denitration in the Glover tower.

For practical purposes the limit of concentration in the Glover tower may be taken as 62° B. The tower will readily concentrate to that strength more acid than is required for the Gay-Lussacs; in fact, the entire make of the chambers. The commercial Glover tower acid is generally of 60° B. strength. The advantages of the tower method of concentration consist in the recovery of a large percentage of nitrous oxide and the raising of the whole output of chamber acid from 50° B. to 62° B., the steam thereby produced being returned to the lead chambers, where steam is needed, and the cost of a part of the water evaporation for that purpose being saved. The only drawback to the method is the impurity of the acid produced and the difficulty encountered in its purification by precipitation at such density. The impure Glover tower acid is sufficient, however, for many chemical uses and it can be concentrated further and purified by special processes.

Plant Required.—The plant required for the manufacture of sulphuric acid of 60° B. strength from zinc blende ore consists, therefore, of muffle roasting furnaces, a system of dust chambers for settling the dust carried over from the furnaces and flues leading from the dust chambers to the Glover tower, a series of lead chambers through which the gas from the Glover tower passes successively, and finally a series of Gay-Lussac towers.

The zinc smelter who proposes to undertake the manufacture of sulphuric acid is obliged to substitute muffle furnaces for his reverberatories. As previously pointed out in the chapter on blende roasting, the cost of roasting in muffle furnaces need not be necessarily any more than roasting in reverberatories. While an elaborate set of dust chambers connected with the roasting furnaces is not essential, they are to be recommended since flue dust is troublesome in the Glover towers, but in arranging a system of dust chambers and the flues leading from them to the tower, care must be taken that the gas be not cooled below the temperature required for the most efficient operation of the Glover tower. The draught through the sulphuric acid system is best regulated by means of fans.

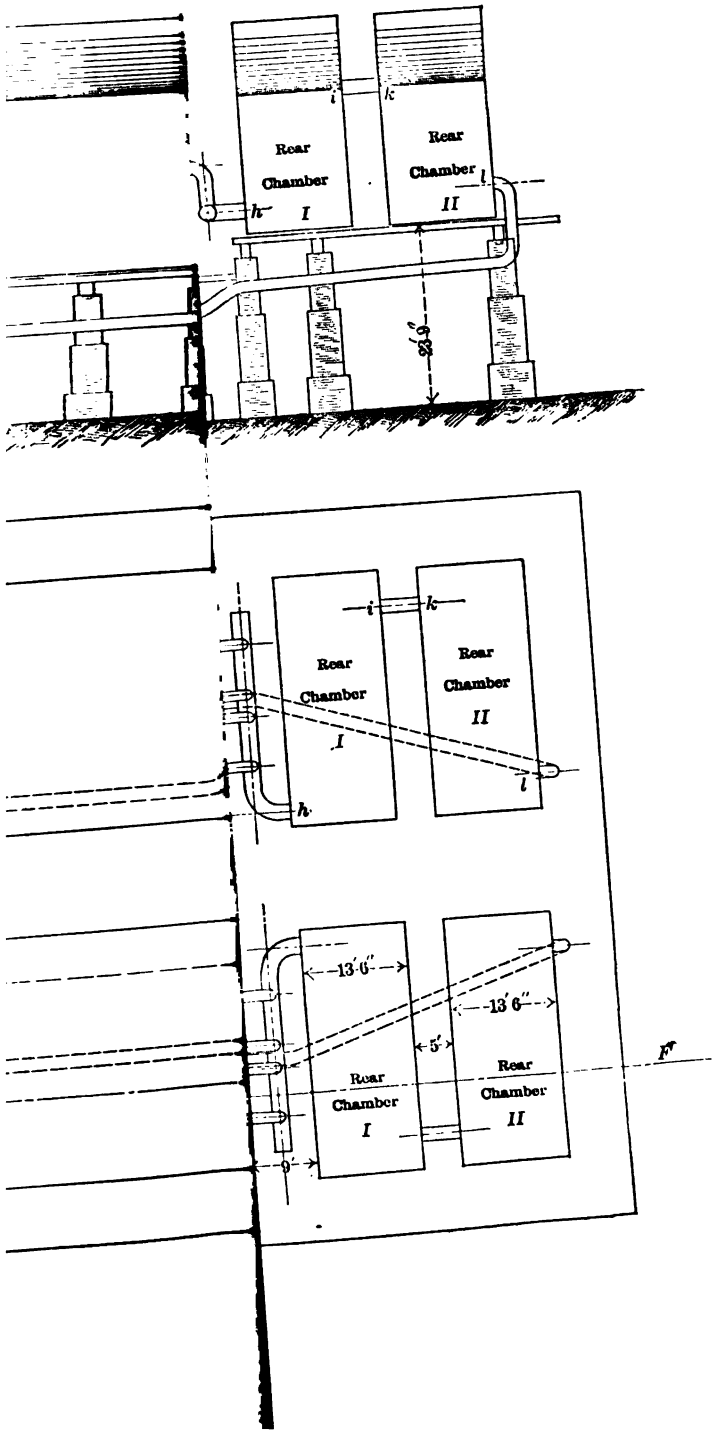
The size of the lead chambers is governed by the quantity of gas th to be condensed in them and its tenor in sulphurous anhydride. Acco to Mr. Falding (loc. cit.), with a suitable arrangement of chambers a regular flow of gas of proper tenor in SO_2, from 11 to 12 cu. ft. of cha space per pound of sulphur in the form of pyrites burned per 24 hou sufficient; with the addition of Lunge columns the chamber space m reduced to 8 or 9 cu. ft. per pound of sulphur. In most existing p however, the proportion of chamber space is considerably greater th indicated by those figures, and an allowance of 15 cu. ft. of chamber per pound of sulphur in the form of pyrites and 16 cu. ft. in the fo blende, no Lunge columns being employed, would be considered good tice. The volume of the gases obtained in roasting blende being som greater (in the ratio of 800 : 840) than those from the roasting of p under the same conditions (vide p. 40) the chamber space must be inc proportionately.

General Arrangement.—The general arrangement of a sulphuric plant is illustrated in the accompanying drawings (reproduced from a by Robert Hasenclever in *Die Chemische Industrie,* of January 15, 189 the installation at Stolberg, Germany, where the sulphurous anhydrid is produced by roasting blende. The plant comprises six double H clever furnaces, which have an aggregate capacity for roasting 48,000 kg. of ore per day. The volume of the lead chambers is 7250 c (256,026 cu. ft.) per system. An average output of 40,640 kg. of a 60° B. has been made with this plant, the roast gases assaying 9% by volume.

It is to be remarked that this plant is operated by natural draugh use of mechanical exhausters being rare in Europe, though they are the common practice in the United States. The roasting furnaces a at the unusual distance of 11 m. (37 ft.) apart for the greater co of the men. The lead chambers have the six-side section, which was fa by Herr Hasenclever on the grounds of economy of lead in constru better utilization of the chamber space "because of fewer dead-corners adaptability to the ordinary pitch roof, permitting a construction affords easy access to the top of the chamber without making the r the building especially high for that purpose. The chambers are ported from the ground in such a way that a large clear space is ava for storage tanks, acid concentration apparatus, etc. The building e ing the chambers consists of a substantial timber frame sheathed boards and a roof covering of tile.

The chambers are divided into two systems, each of which is serv

n on line A.B



three furnaces. In order to control the draught perfectly each furnace is sucked by a separate flue, the gas from the three being united only when it reaches the Glover tower. The flues are of large dimensions and the velocity of the gas through them is slow, giving the dust ample opportunity to settle. There is in the first place a dust chamber built in the upper part of each single furnace. In that the gas, then at a temperature of

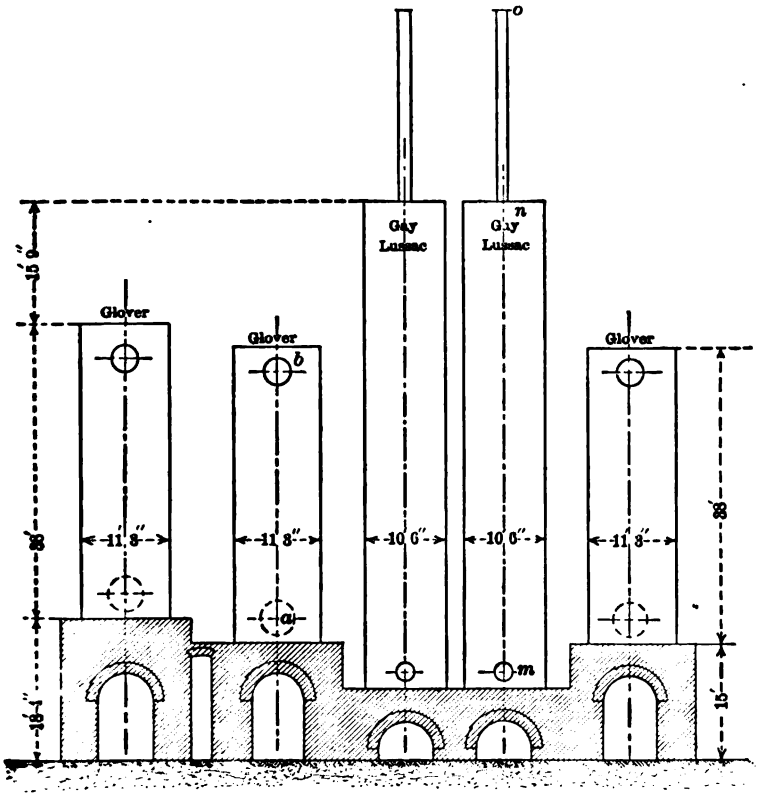


FIG. 131. BLENDE ROASTING AND SULPHURIC ACID PLANT.
Elevation of towers. Scale, 1 : 300.

580° C., moves at the rate of only 0·138 m. per second. In the canals leading from the furnaces the velocity varies from 0·5 to 0·65 m. per second. The gases reach the Glover tower at a temperature of 300° to 350° C. The low velocities registered in the flues of the dimensions shown in the accompanying engravings are due of course to the unusual richness and consequently small volume of the gas, which assaying 9% SO_2 is in quantity

only one half what it is when the gas contains 4·5% SO_2—a figure which is not uncommon in practice.

Arrangement of Chambers.—It is not merely sufficient to provide sufficient chamber capacity, but the latter must be arranged according to the zones of chemical activity. It is obvious that if there were only one long chamber the most intense chemical activity would occur at the end into which the fresh gas from the Glover tower is admitted and in proceeding toward the other end the action would become more and more sluggish, because of the increasing dilution of the gas, until it would cease entirely.

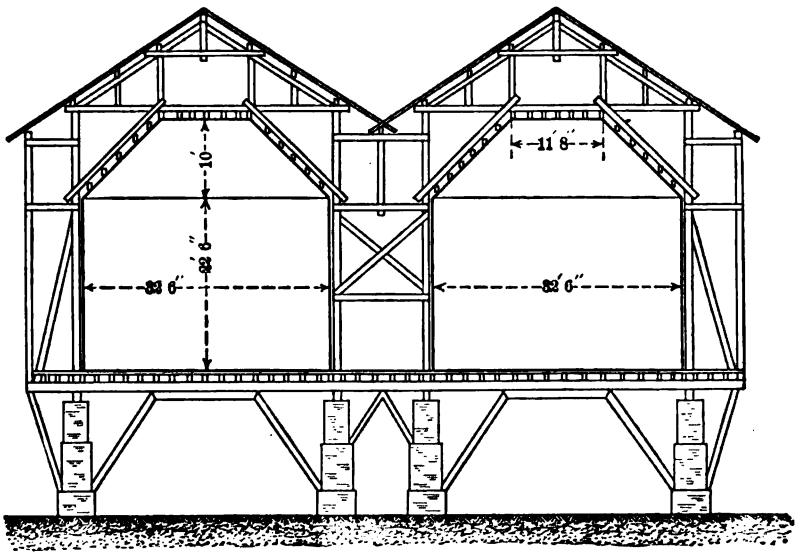


FIG. 132. BLENDE ROASTING AND SULPHURIC ACID PLANT.
Transverse vertical section through chambers. Scale, 1 : 300.

An installation of a single long chamber, even if it had the requisite volume, would be very inefficient. Practically it is found that the greatest formation of acid takes place within a comparatively short distance, generally less than 100 ft., after which the chemical activity diminishes sharply; up to that point fully 70% of the sulphurous anhydride will have usually been converted into acid.

It is customary therefore to divide the chamber system into a series of three or more, which correspond to zones of chemical activity. By causing the gases to pass through the comparatively small connecting pipe from one to the next, they are mixed and the intensity of the process is greatly in-

creased for a time. Usually the division is made into only three chambers, of which the length is in the ratio of approximately 5 : 3 : 1. In the first, together with the Glover tower, about 70% of the entire formation of acid takes place; in the second about 25%; in the third the last of the acid is made and the nitrogen oxides are prepared for absorption in the Gay-Lussac towers. Mr. Falding recommends that the first chamber be made not over 80 ft. in length, the following chambers being in proportion. In large systems the practicable cross-sectional dimensions will not afford a sufficient volume with such lengths, and it is the common practice to install four, five or six chambers in the series and make them of great length, even up to 200 ft.; in such a system "the chemical activity becomes less and

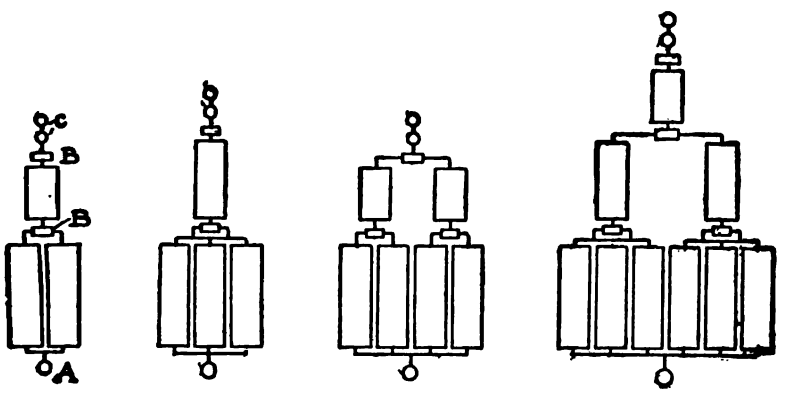


FIG. 133. DIAGRAM OF CHAMBER SUBDIVISION.
A, Glover tower. *B*, Intermediate columns. *C*, Gay-Lussac tower.

less, except where at the beginning of each new chamber it is more or less stimulated, until finally by sheer space and distance the sluggish process is completed, always in presence of the danger of loss of niter and great corrosion of the final chamber and connections, either by formation of nitric acid or by higher oxidation of the N_2O_3, which accidents do in fact occur."

In order to secure the requisite chamber volume for a large plant without increasing the length of the individual chambers Mr. Falding makes the rational suggestion that the first, second and third be built in multiple and grouped abreast, the gas being divided between them, and the number of chambers in each group being proportioned to the volume of gas and other conditions—an arrangement which he states he has found in every way satisfactory in practice. Such an arrangement is illustrated in the accompanying engraving. The proportion between the first, second and third

chambers of the series is approximately 4 : 2 : 1. To make the division practically, divide the total cubic feet required by the sectional area that has been determined upon, which will give the total length of the chambers. Then proportion the length as 4 : 2 : 1 :, and if the cubic capacity is so large that the first chamber will be more than 100 ft. long, divide it into two or more chambers. For example, supposing the total volume required were 100,000 cu. ft. and the sectional area to be employed were 520 sq. ft. (26×20 ft.) then 100,000÷520=approximately 193 ft. and the system would be divided into three chambers about 110 ft., 55 ft. and 30 ft. respectively. But if 400,000 cu. ft. were required and the sectional area were

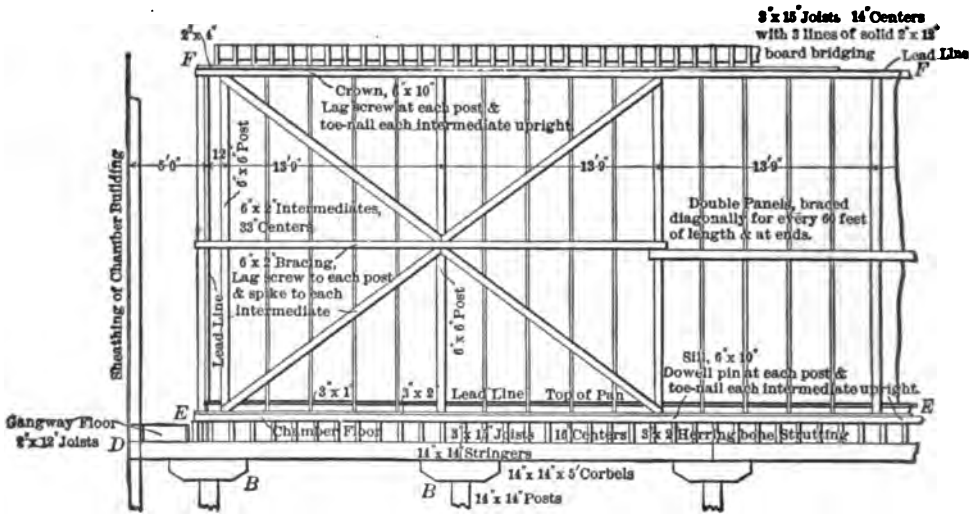


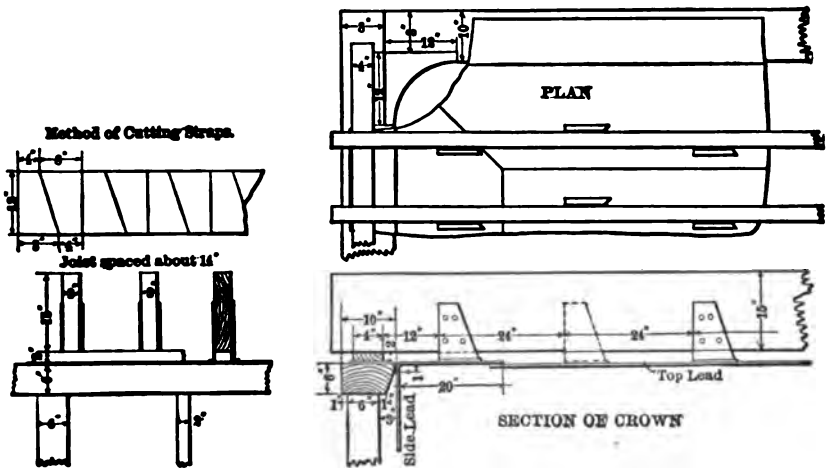
FIG. 134. ELEVATION OF CHAMBERS.
Scale, ¼ in. = 8 ft.

to be 720 sq. ft., the total length of chamber would have to be 400,000 ÷720=approximately 550 ft. and the ratio 4 : 2 : 1 would make the leading chamber 320 ft. long, the second 160 ft. and the third 75 ft. Those lengths would be much too long for the first and second chambers, so the first would be divided into four of 80 ft. length and the second into two of 80 ft.

The sectional area that may be given a sulphuric acid chamber depends upon the size of the plant and structural considerations. In the United States they are generally made rectangular. Dimensions of 24×20 ft., 26×20 ft., 27×24 ft., 28×24 ft., 30×20 ft. and 30×24 ft. are common.

Details of Chamber Construction.—A sulphuric acid chamber is essenti-

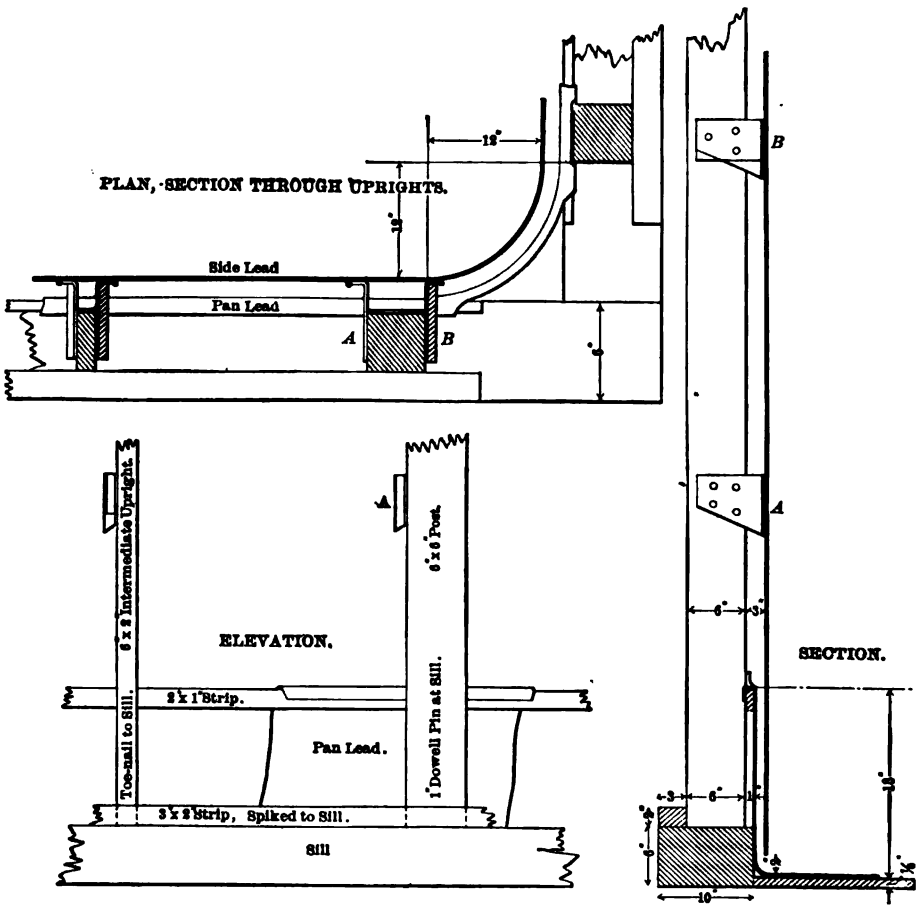
ally a huge box of sheet lead, which is supported by an external framework of timber. The floor of the chamber must be sufficiently high above the ground to give easy access to its under side. The sheet lead should be of the variety known as "chemical." Sheet lead weighing 6 lb. per sq. ft. is commonly employed, but sheets as light as 4 lb. and as heavy as 8 lb. are sometimes used. The heavier the lead the greater is the first cost of the chambers, but the less is the cost for repairs and renewals in the operating account. In comparing bids for the construction of chambers special attention should be paid to the weight of the lead that it is proposed to use, since contracting engineers sometimes make proportionately low bids by



FIGS. 135 TO 138. DETAILS OF CHAMBERS.
Scale, 1 in. = 3 ft.

planning to substitute a light weight of lead for what should really be used. The cost of the lead is a very large item in the erection of chambers and the expense of the latter varies naturally a good deal according to the price for the metal. The quantity of lead required for the chambers is approximately from 1·3 to 1·4 times the weight calculated on the basis of their surface; for the towers 0·2 to 0·3 times the calculated weight of chamber surface; and for other purposes 0·1 to 0·2; or for the whole plant 1·7 to 1·8. The lead sheets are united by burning in the ordinary manner, and are fastened to the timber-frame by means of straps burned to the outer surface. The chamber lead is somewhat corroded by the process of acid manufacture, but not to a great extent, and in dismantling an old acid plant something like 90% is recovered.

The bottom of the chamber is made in the form of a pan, within which the curtains or sides of the chamber extend to such a depth that the opening is sealed by the acid collected in the pan. The acid outlets are arranged



FIGS. 139 TO 141. DETAILS OF CHAMBERS.

For chambers 20 ft. high burn four straps on alternate sides of each post and intermediate upright at heights respectively of 3 ft. 4 in., 6 ft. 8 in., 10 ft., and 13 ft. 4 in. from chamber bottom. *A* shows strap unfinished. Use four leaded nails 1·5 in. long and turn over 1 in. at the wider end for burning to side lead. Finish by spiking on a 0·75 in. board, 4×6·5 in., as shown at *B*. The lead straps are cut from a strip of sheet, 9 in. wide, into trapezoids 3 in. wide on the short side and 6 in. on the long side, whereby 9×9 in. of lead cuts into two straps. The method of cutting the lead straps is shown in Fig. 135.

in such a way as to keep a layer of 3 to 4 in. of cool acid always in the pan, which prevents any scouring action on the bottom. The steam is

admitted through pipes entering the top of the chambers at points about 25 ft. apart, beginning 25 ft. from the front end of the chamber. In narrow chambers the openings are made on the center line; in wide chambers two holes may be placed abreast on each side of the center line. Sometimes the steam is admitted at the connections between chambers in such a way as to act as an injector in case of insufficient draught. The steam pressure employed averages about 20 lb., ranging from 10 to 40 lb. Acid chambers are also fitted with drips for drawing off samples of acid for testing, with thermometers for determining the temperature and other necessary devices for the control of the process. The details of construction are

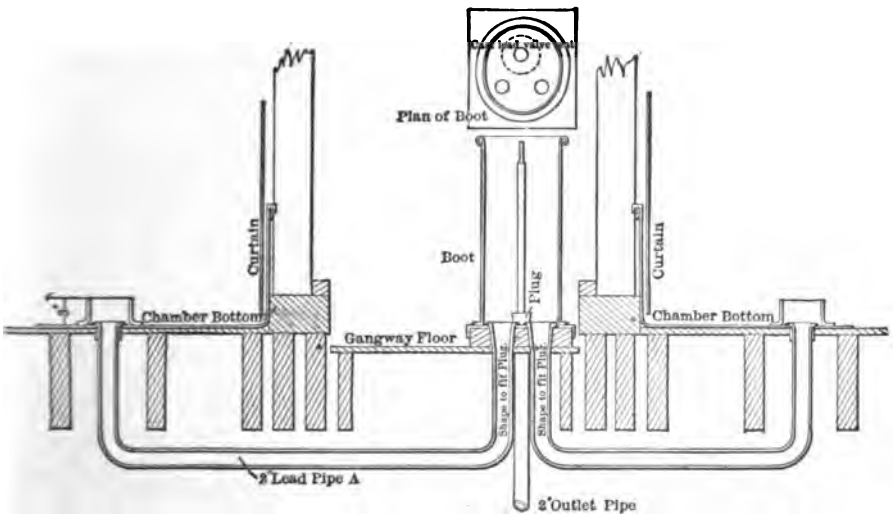
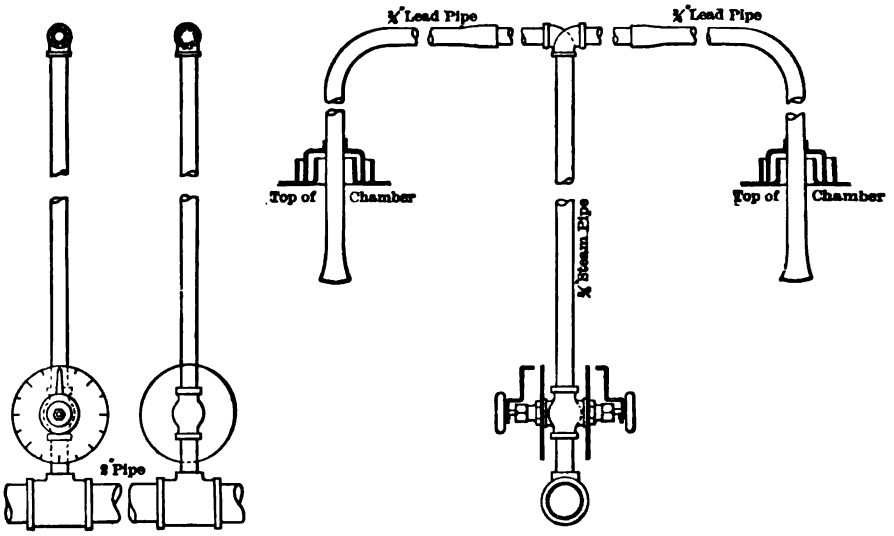


FIG. 142. DETAILS OF CHAMBERS; ACID CONNECTIONS.
Scale, 1 in. = 2½ ft.

illustrated in the accompanying engravings from drawings by Mr. F. J. Falding, reproduced from his paper in *The Mineral Industry,* Vol. VII, to whom are due also the engravings of the Glover and Gay-Lussac towers.

Glover Tower Design.—The Glover tower is a shaft of heavy sheet lead, properly stayed and supported on the outside, lined with about 9 in. of refractory material, which may be chemical brick, but preferably lava; and packed with chemical brick or quartz resting on a suitable grill. The height and diameter of the tower are proportioned to the volume of the gas which is to pass through it, the means taken to create the draught, and upon whether high concentration or high oxidation be desired. In any case,

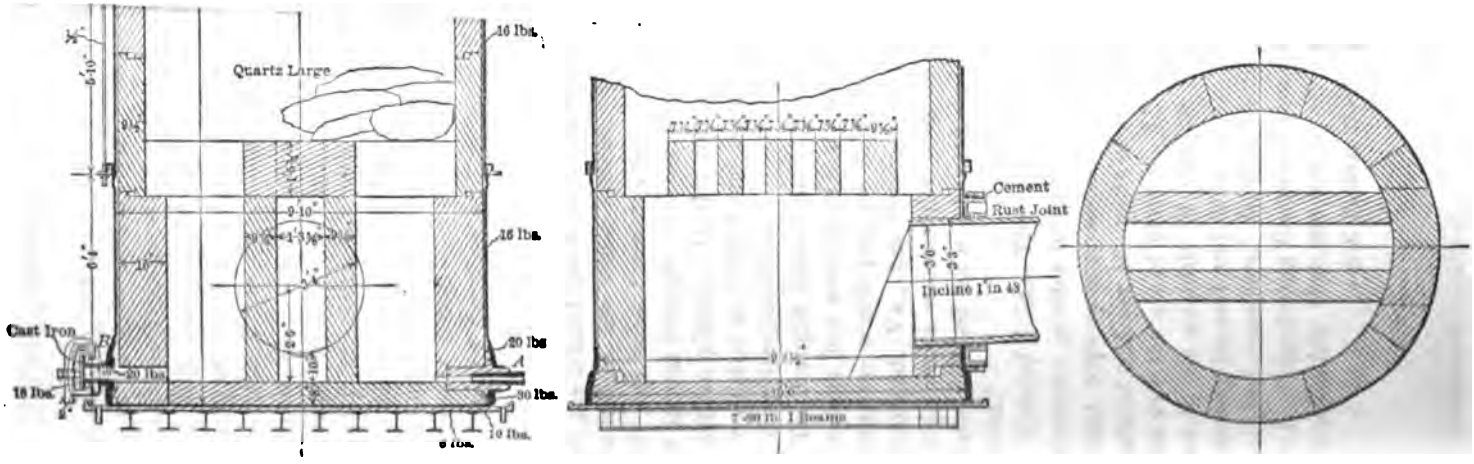
the proportions must be such that the heat in the sulphurous gas delivered to the tower will be absorbed and prevented from passing to the chambers, but the exit gases must be sufficiently hot to prevent condensation of any considerable proportion of the steam formed in the lower part of the tower. Mr. F. J. Falding (loc. cit.) states that towers of greater height than 18 to 25 ft. (according to the volume of gas they are designed to handle) are unnecessary; although "higher towers may on account of the cooling of the gas produce more acid owing to condensation, it can only be at the expense of the concentrating efficiency of the tower." Concentration, it must be remembered, is a more important function of the Glover tower than con-



FIGS. 143 AND 144. DETAILS OF STEAMPIPE CONNECTIONS FOR CHAMBERS.
Scale, 1 in. = 1 ft.

densation, which should take place in the chambers. Formerly towers 40 to 50 ft. in height were used. In any case the packing must be arranged in such a way that there will not be an excessive obstruction to the draught. The height of the packing in a tower from 18 to 25 ft. high will be 10 to 17 ft.

Design of the Gay-Lussac Tower.—The Gay-Lussac tower is a slender shaft of 35 to 50 ft. in height and 6 to 10 ft. in diameter. When a greater height than 50 ft. is necessary it is advisable to divide it between two or more towers. The shaft is supported by an external framework of timber, and is lined with chemical brick. It is packed with quartz supported on a grill of chemical brick. Chemical brick or perforated tiling also form a


Quartz Large
16 lbs.
16 lbs.
20 lbs
30 lbs.
10 lbs.
Cast Iron
Cement
Rust Joint
Incline 1 in 43

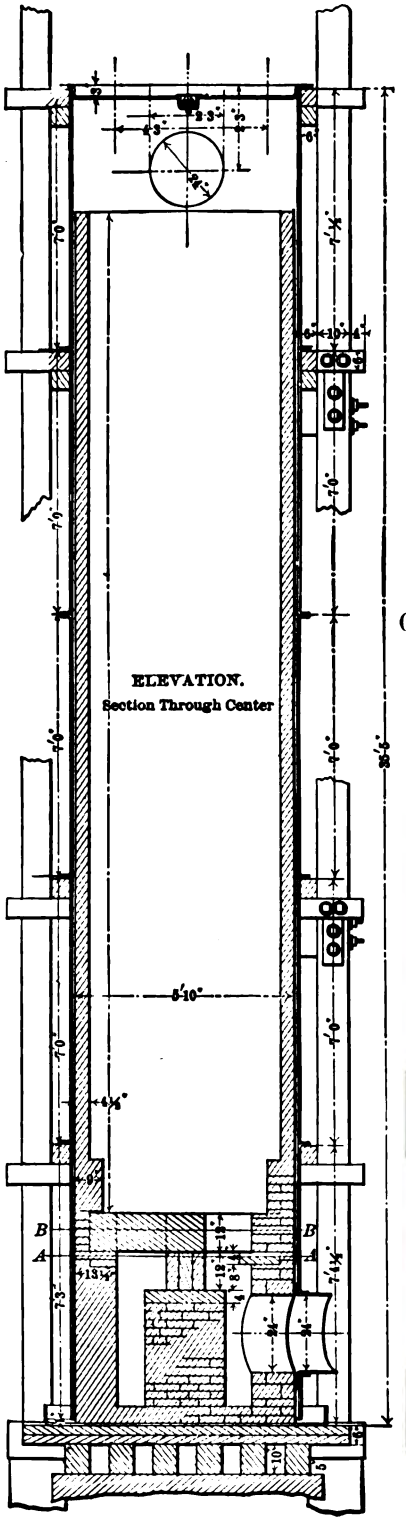
good packing, but coke which used to be employed commonly, and is still used to a considerable extent, is not to be recommended, since it breaks down and cements together so that the passage of the gas is greatly interfered with, and also it reacts chemically with the nitrogen compounds, forming oxides which are not absorbed. Quartz packing, put in with the large pieces at the bottom and the small ones at the top, is one of the best materials, while the balls, cylinders, troughs, star-plates and other elaborate packings, used in Germany, which can be placed in position with great accuracy, are claimed to be highly efficient. The Gay-Lussac tower should be placed in a cool and easily accessible position and requires no protection from the weather, save from the direct rays of the sun.

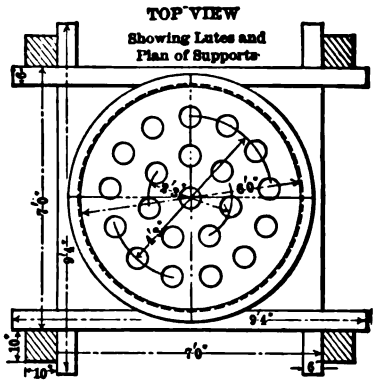
Draught Fans.—In the ordinary sulphuric acid plant the draught on the roasting furnaces and through the entire system is produced by the vacuum caused by the condensation of the gases into sulphuric acid and the contraction in volume due to cooling, by the siphon-like action of the cool gas behind the Glover tower and by a chimney or steam inspirator at the exit of the whole system. The Matthiessen & Hegeler Zinc Co. in 1885 first substituted mechanical draught induced by a suitable exhauster for natural draught, and that system is now generally employed in modern American plants. In this way the draught through the whole system can be regulated as desired and is uninterfered with by atmospheric conditions or other irregularities.

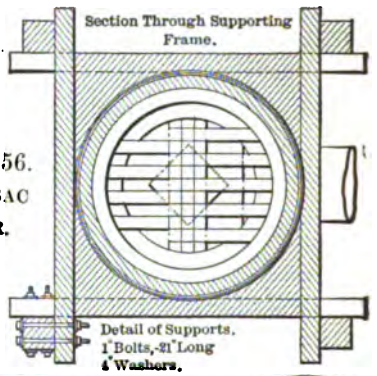
The draught fan is usually placed between the Glover tower and the lead chambers, but sometimes it is installed immediately after the roasting furnaces, for which purpose A. P. O'Brien, of the Richmond Guano Co., of Richmond, Va., has designed an exhaust fan of cast iron construction which works successfully with sulphurous gas of 540° C. (1000° F.). Water is used as a lubricant instead of oil and the journals are water cooled.[1]

Yield of Acid.—Theoretically 100 parts of sulphur burned in the form of brimstone, pyrites or blende, should make 306·25 parts of H_2SO_4. The actual recovery approaches frequently close to the theoretical, yields of 300 parts of acid per 100 parts of sulphur burned having been recorded in practice with the use of pyrites. These ratios, it should be borne in mind, refer not to the quantity of sulphur contained originally in the ore, but the quantity burned and delivered in the gas which is led to the Glover tower and lead chambers. In practice the yield is generally less than the ratio 3:1. Considering the production of acid from zinc blende, it may be assumed safely that one ton or ore containing 60% Zn and approximately

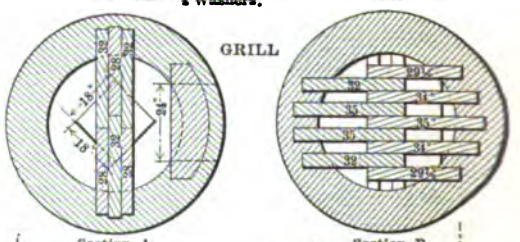
[1] F. J. Falding, The Mineral Industry, IX, 621.

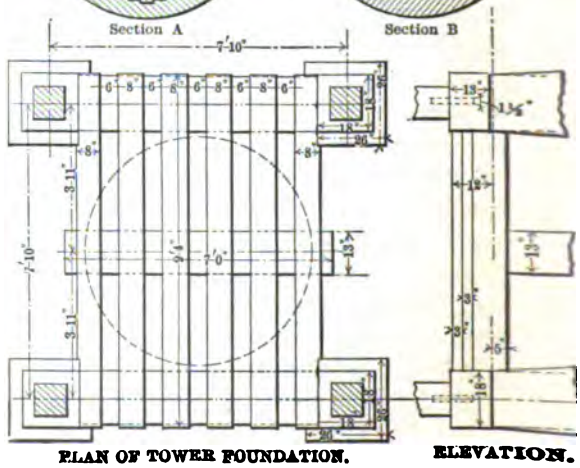






FIGS. 150 TO 156. GAY-LUSSAC TOWER.





30% sulphur, will yield one ton of acid of 60° B. if 29 units of the sulphur be burned, and at least as much as that would be necessarily burned off, because even then the tenor of sulphur in the roasted ore would be approximately at the limit that could be tolerated for zinc smelting. Assuming then that 29 units of sulphur, equivalent to 580 lb., be burned, the ratio of acid production would be as follows: One ton of 60° B. acid contains $2000 \times 0{\cdot}77605 = 1552{\cdot}1$ lb. of H_2SO_4, and $1552{\cdot}1 \div 580 = 267{\cdot}6$; i.e., 267·6 parts of H_2SO_4 has been reckoned per 100 parts of sulphur burned, which is a moderate estimate. If 300 parts of H_2SO_4 were made per 100 parts of sulphur burned, the yield of 60° B. acid would be 2240 lb., or 1·12 tons.

The tenor of H_2SO_4 in commercial acid of various degrees of strength may be found from tables contained in the several treatises previously referred to. The data of that nature required for the present discussion of the subject is summarized as follows:

Acid of 66° U. S. Beaumé scale contains						93·500%	H_2SO_4.	
" " 60° "						77·605%	"	
" " 50° "						62·177%	"	
2,000 lbs. of 66° acid contain						1,870·0 lbs.	H_2SO_4.	
" " " 60° "						1,552·1 "	"	
" " " 50° "						1,243·5 "	"	
1 ton of 66° acid	=	1·20	tons of	60°	acid			
1 " " 60° "	=	1·25	" "	50°	"			
1 " " 50° "	=	0·80	" "	60°	"			
1 " " 60° "	=	0·83	" "	66°	"			

Cost of Production.—The cost of acid manufacture is a function of the size of the plant to a greater extent than in most branches of metallurgical industry, because after the gases are delivered from the roasting furnaces the materials which enter into the composition of the acid are chiefly in gaseous and liquid form, their movement through flues and pipes being controlled by fans and dampers, pumps and valves, wherefore the same number of men can with a plant of adequate size make 50 tons of acid per day as well as 20 tons. On that account and in view of the active competition which exists at present in the sulphuric acid business, it would probably be inadvisable to plan a sulphuric acid plant of less capacity than 10,000 tons of 60° B. acid per annum. The cost of a sulphuric acid plant of 10,000 tons capacity per annum of Glover tower acid, not including the roasting furnaces, is approximately $60,000, or $6 per ton of annual capacity, when pig lead is worth about 4c. per lb. at New York. Since the cost of the sheet lead for the chambers forms an important part of the total cost, the latter is increased or decreased by a rise or fall in the value of lead. The cost of the plant per ton of capacity decreases also to a certain extent with the size of the plant, wherefore a plant of 30,000 tons capacity would probably cost only about $5 per ton. Sulphuric acid chambers are

generally enclosed in wooden houses, but brick houses are sometimes employed.

As compared with the manufacture of sulphuric acid from brimstone and pyrites, the manufacturer from blende has the advantage of saving the cost of the raw material[1] and the cost of handling and burning it, since his raw material and the cost of roasting it will be charged properly against the product of spelter. Moreover, his general expense account will be to a considerable extent shared by the zinc smelting side of the business.

In making sulphuric acid from blende, assuming that the sulphurous gas be delivered free to the acid department, the cost of production apart from amortization of the plant will include: (1) Interest on working capital; (2) labor; (3) coal; (4) nitrate of soda; (5) miscellaneous supplies; (6) repairs and renewals.

1. General expense: This will include taxes, insurance, interest on working capital, salaries and office expenses and the cost of marketing the product. These will vary considerably under different conditions.

2. Labor: Probably the maximum number of men required for a plant capable of making 30,000 tons of 60° B. acid per annum, would be as follows: One general foreman, four chamber and tower men, four pump men, and four yard men and laborers.

3. Coal: Coal is required for evaporating the water to combine with the sulphuric anhydride and also for power in operating the pumps and fans. One ton (2000 lb.) of acid of 60% B. strength contains 742 lb. of water. Coal is required to evaporate in boilers that quantity, besides which a certain amount of power is required with the result that the total consumption of coal will be probably about 10% of the product of acid.

4. Nitrate of soda: In good practice, with the proper system of Gay-Lussac and Glover towers, the loss of nitrate of soda will be 2·5 to 3% of the weight of the sulphur burned. In the decomposition of the additional niter to make good the loss, a corresponding quantity of sulphuric acid is consumed. If the loss of niter were 3%, on 580 lb. of sulphur it would be 17·4 lb., which would require 20 lb. of H_2SO_4.

5. Miscellaneous supplies, light, etc., will probably come to about 10c. per ton of acid produced.

[1] This is only true, however, when the ore is obtainable at the same price as is established by those zinc smelters who do not recover acid. If the ore price be determined by the competition of smelters who all make acid, its sulphur contents will be paid for in one form or another. The combination of zinc smelting and acid making may also involve disadvantageous freight charges and other costs, which must really be debited against the sulphur in the ore. It will probably be rarely the case that the maker of acid from blende will have an advantage over the maker from pyrites to the full extent of the cost of the pyrites.

6. Repairs and renewals: These may be estimated at 50c. per ton of acid produced, which figure is sufficiently liberal to include the repairs required on the tank cars for distribution of the product.

The total cost of acid recovery, not including general expense or amortization of plant, will probably come to $1·25@$1·50 per ton of 60° B. acid, f. o. b. works.

Tank Cars.—In distributing the acid product to customers, a number of tank cars are required, which as a general thing are the property of the acid manufacturer, the railways making freight rates for the movement of acid in shippers' cars, the quoted rates usually covering the free return of the cars to the acid works. The number of cars required depends upon the quantity of acid that is to be produced and the location of the acid works with reference to the points where the products are consumed.

The tank cars employed for the transportation of acid are similar to those used for carrying petroleum and other liquid products. The tanks are made usually of ⅜ in. steel plates, cylindrical in form. The corrosion of acid of 60° B. strength upon iron and steel is unimportant, and with proper care acid of 50° B. strength may be shipped in such tanks without significant wear and tear. Cars are made generally with tanks of 20, 30 and 40 tons capacity. Under the present conditions of transportation it is inadvisable to order 20-ton cars, since the railway equipment of the country is yearly becoming heavier and cars of 20 tons capacity have been almost entirely abandoned, owing to the fact that they are liable to be crushed or pulled apart between cars of 30 and 40 tons capacity when coupled in the same train. If for any reason it is desirable to have cars of only 20 tons capacity, it is preferable to place a 20-ton tank on a 30-ton car. The cost of a tank car of capacity for 30 tons of 60° B. acid, made of ⅜ in. steel plate, double riveted throughout, equipped with air brakes and M. C. B. automatic couplers, and conforming to all the requirements of the M. C. B. Association and the national laws, is about $985 at the maker's works. A car of the same capacity, but made of ¼ in. steel, costs about $840. A 30-ton car with a double-riveted 20-ton tank costs $940, while the same car with a single riveted tank costs only $790. If 30-ton cars are used, it is most advisable to purchase those with double riveted tanks of ⅜ in. steel, while for 20-ton cars a single riveted tank may be sufficient, the car body and trucks remaining the same in each case.

Catalytic Manufacture of Sulphuric Acid.—At the present time the sulphuric acid industry bids fair to be revolutionized by the successful introduction of catalytic processes. These processes are based on the idea of making the sulphurous anhydride obtained by burning sulphur, or sulphides,

combine directly with oxygen to form sulphuric anhydride, which is then easily absorbed in water, so as to make commercial acid of any desired strength up to pure H_2SO_4. The idea is an old one, and the fact that finely divided platinum in virtue of its power of condensing oxygen induces the union of SO_2 and O into SO_3 has been known for a long time, but all attempts to utilize the reaction for the production of sulphuric acid from a mixture of sulphurous anhydride, air and nitrogen, were failures until recently. "The platinum acted too feebly in the presence of the unavoidably large volume of nitrogen and soon lost its efficacy altogether owing to the accumulation on it of particles of dust from the kiln gases. Chromic oxide and ferric oxide act like platinum, through the transitory formation of their respective sulphates—the gases produced in pyrites kilns include a considerable quantity of ready made sulphuric anhydride—but they also are not available practically for the making of sulphuric acid." [1]

Since the paper above quoted was written, great progress has been made in the development of catalytic processes. In a lecture delivered before the Liverpool section of the Society of Chemical Industry, October 4, 1899, Professor Lunge said that when he wrote for *The Mineral Industry,* Vol. IV, at the end of 1897, approximately 67% was to be considered very good work, the remaining 33% of the sulphurous acid gas having to be sent to the lead chambers together with better gas from other burners; "in lieu of 67% we must *now* speak of 98%."

Catalysis Defined.—Catalysis is defined by Berzelius as a decomposition and new combination produced among the proximate and elementary principles of one or more compounds by virtue of the mere presence of a substance or substances which do not of themselves enter into the reaction. It is believed, however, that bodies which cause catalysis do in some way take part in the chemical reaction involved, though they are in the course of it always brought back to their original condition. Platinum black absorbs 800 times its volume of oxygen from the air and in virtue thereof is a most active oxidizing agent, which in general acts catalytically, because the black after having given up its oxygen to the oxidizable substances present, at once takes up another supply from the atmosphere.[2]

Practical Applications.—If gaseous sulphurous anhydride be heated to a temperature of 380° C. with its theoretical volume of oxygen (or air) and be brought into intimate contact with finely divided platinum, ferric oxide or other catalytic material, an almost complete oxidation to sulphuric anhydride will take place, providing the temperature of the reaction during the

[1] Encyclopedia Britannica, 9th edition, XXII, 636.
[2] Ibid, XIX, 191.

contact is not allowed to rise to the point of dissociation. From 400° to 430° the reaction remains nearly uniformly quantitative, a conversion of 98 to 99% being maintained. Above this, decomposition of the sulphuric anhydride becomes perceptible and at 700° to 750° the conversion is only about 60%. Between 900° and 1000° no action between sulphur dioxide and oxygen can be detected.

The activity of the catalytic agent, at least so far as platinum is concerned, seems to vary directly with its surface exposed. At first platinum wire was used, afterward platinum sponge, platinized pumice, platinized asbestos (Winkler patents) and finally a platinum mass consisting of a crystallized double salt of platinum and a water-soluble sulphate (Grillo-Schroeder, United States patents Nos. 636,924 and 636,925), the claim of the last mentioned process being "A contact body for use in catalytic processes, comprising salt crusts, containing a platinum salt distributed through a mass of one or more of the other soluble salts which serve as a carrier or vehicle therefor." By the use of such a contact substance the quantity of platinum required for the process has been reduced to less than one half of that necessary when platinized asbestos is employed, probably in part because of the extremely fine division of the platinum and in part because the associated sulphate of itself possesses catalytic properties.[1]

Essential Conditions.—In the production of sulphuric anhydride by catalysis it is essential that the mixture of air and sulphurous anhydride be free from all impurities other than nitrogen. Arsenic compounds, silica and flue dust especially interfere with the reaction, probably by coating the contact mass and reducing the exposed surface of the active substance. In the Grillo-Schroeder patents it is claimed, however, that in case of weakening of the activity of the contact mass by any carelessness in purification of the gas, the contact substance because of its complete solubility can be easily regenerated and practically without loss of platinum.

Among the other difficulties encountered originally in the development of catalytic processes was the thermo-chemical one that sulphuric anhydride is decomposed at a certain high temperature into sulphurous anhydride and oxygen, and the combination of sulphurous anhydride and oxygen being itself a strongly exothermic reaction, it was difficult to control the temperature within the narrow limits (vide supra) wherein the sulphuric anhydride produced is stable. Inasmuch as the combination occurs only at high temperatures the mixture of gases must first be heated before they will combine, and the heat of the combination, added to the initial heat, produced so high a temperature that the sulphuric anhydride itself was decomposed,

[1] F. J. Falding, The Mineral Industry, VIII, 588.

while the apparatus in which the combination was effected was destroyed rapidly by oxidation and the contact substance was deteriorated.

The thermo-chemical difficulty was overcome in the process of the Badischen Anilin und Soda Fabrik by removing the excess of heat from the contact substance and the apparatus by a readily controllable means of external cooling. Such a device is substantially all that is covered in the original Badischen patents.

The best absorption medium for the sulphuric anhydride has been found to be acid containing 97 to 98% of H_2SO_4; this is better than either water or acid of any other strength. A single absorption vessel is sufficient for even a very rapid stream of gas, if the strength of the absorbing medium be kept constant by the regular inflow of water and outflow of the sulphuric acid that is formed.

Present Development of Catalytic Processes.—In the present stage of development there are at least four catalytic processes which are in use by various concerns. One of the more important of them appears to be the Grillo-Schroeder process, the American rights to which have been acquired by the New Jersey Zinc Co. The Mineral Point Zinc Co., a branch of the New Jersey Zinc Co., is now actually engaged in the catalytic manufacture of sulphuric acid at its works at Mineral Point, Wis. Other concerns in the United States which have applied the catalytic process are the Dupont Powder Co., Repauno Chemical Co., Forcite Powder Co., Peyton Chemical Works and Harrison Bros. In Europe catalytic sulphuric acid is made at the zinc works at Hamborn-Neumühl, Westphalia, and at the Lazyhütte, Upper Silesia, besides at various chemical works in England, France, Germany, Italy and Russia.

F. J. Falding in *The Mineral Industry*, VIII, 587, stated that the cost of production of 50° B. acid by this process, compared with its cost when made with a modern set of chambers, shows a fair margin of profit in favor of the contact process, and for acids above 60° B. the margin increases rapidly with the strength. In addition to these advantages, the smaller cost of plant and less ground space required, are decidedly in favor of the contact process. Moreover, in the contact process the same plant which produces 50° B. acid can with very slight modification be made to produce 66° B. acid, or even the monohydrate itself or any of the fuming or Nordhausen acids or the actual anhydride, without additional labor or cost. Three of the concerns which are promoting catalytic processes are obtaining and are willing to guarantee a total yield of 95 to 98% of the sulphur as acid which is theoretically possible.

F. Lüty and H. H. Niedenführ consider, however, that the superiority of

the catalytic process has not yet been proved, except for the manufacture of highly concentrated acid. They compute that the chamber process with all of the modern improvements can be made to produce acid of 50° B. at considerably lower cost than the catalytic process, while with the Kessler system of concentration the two processes are about on equal terms in the manufacture of 66° B. acid. In making the higher grades, however, the catalytic process is decidedly superior.[1]

[1] Preëminent among the recent contributions to the literature of the present development of the catalytic process is the paper by Dr. Rudolf Knietsch, himself the chief inventor of the Badischen method, in the Berichte der Deutschen Chemischen Gesellschaft, XXXIV, xvii, 4069 to 4115, Dec. 28, 1901, of which there is a translation in *The Mineral Industry*, X, 605 to 635. Other valuable papers have been published by F. Lüty and H. H. Niedenführ, Zts. f. angew. Chem., XV, xi, 242 to 257, March 18, 1902; Theo. Meyer, ibid., 1901, XIV, 1; L. Pierron, Revue de Chimie Industrielle, April, 1902; and Dr. Steger, Zts. f. d. Berg.- Hütten.- u. Salinenwesen im Preuss. Staate, L, iii, 506 to 530.

VI.

GENERAL PRINCIPLES OF ZINC DISTILLATION.

The zinc in an ore having been converted into oxide by calcination or roasting, the recovery of the metal is accomplished by virtue of the fact that the oxide is reducible by carbon or carbon monoxide, or both, when heated to a high temperature. The precise temperature at which the reduction takes place is somewhat uncertain. According to O. Boudouard, experiments showed a slight development of gas at 800° C., but after heating for some time this ceased completely and a regular reaction did not begin until a temperature of 1125° to 1150° was attained.[1] E. Prost found that the reduction of pure zinc oxide took place at 1075° C.[2] According to R. C. Schuepphaus the reduction begins at 910°, but does not become active until a considerably higher temperature is reached.[3] Hempel also states that reduction begins under the boiling point of zinc (920°).[4] Stahlschmidt, in a series of experiments undertaken at the instance of Robert Hasenclever to investigate the causes of loss of zinc in blende roasting, found[5] that zinc oxide was to some extent reduced when a stream of carbon monoxide was passed over it at "medium red heat," which although vague and doubtful as to the color determination nevertheless indicates a reduction at comparatively low temperature.[6] Irrespective of the temperature at which reduction may begin, however, practically a temperature of about 1300° C. is required. The metal is reduced, therefore, in the form of vapor, which is subsequently liquefied by cooling; in practice, by cooling to a point between 415° C. and 550° C. According to Hempel, the vapor formed by the reduction of ZnO in a blast furnace was not condensed from the gases accompanying it until the temperature had fallen below 470° C. Zinc silicate, Zn_2SiO_4, or $2ZnO, SiO_2$, either anhydrous (willemite) or hydrated

[1] Ann. de Chim. et de Phys., (7), XXIV, 76.

[2] Bull. de l'Assoc. Belge des Chimistes, X, vi, 246 to 263; Journ. Soc. Chem. Ind., 1896, p. 813.

[3] Journ. Soc. Chem. Ind., Nov. 30, 1899, p. 987.

[4] Berg- u. Hüttenm. Ztg., 1893, Nos. 41 and 42.

[5] Ibid, 1875, p. 69.

[6] Full red heat corresponds to about 750° C.; light red, 850° C. The temperature of 1100° C. would be between light yellow and white.

(hemimorphite) is reduced by carbon like zinc oxide, but more difficultly.

Limitations of Zinc Distillation.—Notwithstanding the simplicity of the distillation process in theory its practical execution is attended by so many inherent difficulties that developments on the lines that have been followed in the metallurgy of lead and copper, especially the introduction of furnaces of large capacity and labor saving appliances, have been impossible, and consequently the metallurgy of zinc remains to-day essentially the same as at the beginning. Chief among these difficulties is the ease with which zinc vapor can be oxidized by carbonic dioxide and the necessity of producing a gas of high tenor in zinc; this limits practically the vessels in which the reduction can be effected to retorts or muffles of comparatively small capacity, wherein an atmosphere of carbonic monoxide can be maintained. This limitation has been demonstrated by the experience of nearly a century. All of the numerous attempts to carry out the reduction in reverberatory and blast furnaces have failed. The former give only zinc oxide, while the gases from the latter are so dilute that the zinc distilled can only be condensed to a troublesome powder. In practice, therefore, the zinc oxide is mixed with fine coal and heated to the requisite temperature in cylindrical or muffle shape retorts, from which the zinc vapor is expelled by its own tension and condensed in pipes protruding from the furnace.

Thermochemical Consideration of the Reactions Between Zinc Oxide and Carbon.—The precise reactions which take place in the retort are unknown. It is uncertain whether the reduction of zinc oxide is effected by carbon or carbon monoxide, and if the former, whether one molecule of carbon reduces one of zinc oxide or two. The reactions which may occur, together with the number of heat units absorbed by each, are as follows:

$$\text{I.}\quad ZnO + C = Zn + CO.$$
$$-86{,}000 \qquad +29{,}000 = -57{,}000.$$

$$\text{II.}\quad 2ZnO + C = 2Zn + CO_2.$$
$$-172{,}000 \qquad +97{,}000 = -75{,}000.$$

$$\text{III.}\quad ZnO + CO = Zn + CO_2.$$
$$-86{,}000 - 29{,}000 \qquad +97{,}000 = -18{,}000.$$

$$\text{IV.}\quad CO_2 + C = 2CO.$$
$$-97{,}000 \qquad +58{,}000 = -39{,}000.$$

Each of the above equations is endothermic, i.e., accompanied by an absorption of heat, and, according to Berthelot's law, would be impossible if heat external to the chemical system were not applied. The second equation absorbs the largest quantity of heat and consequently is the least probable. The third equation absorbs the least quantity of heat, but the carbonic anhydride formed by it is reduced by glowing carbon to carbon

monoxide, which is one of the fundamental reactions in all metallurgy, and that reduction is accompanied by the absorption of 39,000 heat units, so that the net result is the same as in equation I.

Roberts-Austen suggests[1] that possibly the carbon under the influence of external heat loses its condensed form and assumes one analogous to that in which it exists in carbon monoxide, so that during the passage from the system ZnO+C to Zn+CO all the heat is absorbed. The part played by the external heat is two fold: It furnishes the energy necessary to depolymerize the carbon, i.e., simplifying its atomic constitution so that it can combine with the oxygen of the CO_2 with evolution of more heat than from carbon in its normal condition, and it may also modify the specific heats and heats of formation of the bodies in presence of each other; while dissociation of carbonic dioxide, setting free oxygen in a specially favorable condition to combine with the carbon and so form carbon monoxide, may also play an important part. Depolymerization of the carbon may put it into such a molecular condition that the heat of formation of carbonic dioxide from it is 117,000 instead of 97,000; this is not an extravagant supposition since it has been estimated that if carbon could be burned in the gaseous state the heat of formation of carbonic dioxide would be not less than 136,000 calories.

Referring especially to reactions II and III, the passage from one to the other may be effected by the presence of even a minute quantity of air entangled in the mixture of ore and carbon in the retort. First, the oxygen in the air may form with the carbon some carbonic dioxide and then some carbonic monoxide, which reacts on the oxide of zinc. The carbonic dioxide, which is the product of this reaction, is again converted by the excess of carbon present into carbonic monoxide. The carbonic dioxide yields double its volume of carbonic monoxide; hence the accumulation of the latter is very rapid, even though its production originated from only a trace of air. A state of equilibrium between the zinc oxide and carbonic dioxide does not become established, because carbonic dioxide is dissociated at the high temperature which exists.

Chemists have differed as to whether the reduction is effected by carbon, or carbon monoxide, or both. It used to be held that it could not be done by the monoxide, but Boudouard after a series of experiments concluded that carbon does not act directly on zinc oxide, but is transformed into monoxide, which effects the reduction.[2] Stahlschmidt in experiments previously referred to found that monoxide passed over zinc oxide reduced

[1] Introduction to the Study of Metallurgy, London, 4th Edition, 1898, p. 340.
[2] Ann. de Chim. et de Phys. (7), XXIV, 76.

metal, which he condensed. This subject has been studied recently from a physical-chemical standpoint by G. Bodländer in a remarkable paper, in which he analyzes various reduction reactions, including that of zinc oxide.[1] He states that the heats of formation do not vary significantly at the high temperature at which this reaction takes place, the reduction of zinc oxide by carbon, with formation of carbon monoxide, still requiring 55,500 calories at 1000° C. Inasmuch as the reaction takes place, the free energy which disappears in the formation of the carbon monoxide must be greater than in the formation of zinc oxide. At absolute zero, where the heat of formation and the energy of formation are the same, the formation energy of zinc oxide is three times as great as that of carbon monoxide. With increase of temperature this relation changes, the formation energy of the carbon monoxide increasing while that of zinc oxide decreases; at the point where they are equal the reaction takes place. In chemical reactions not all of the energy as measured in heat units is capable of performing chemical work; some is consumed in performing certain physical work, and the free energy to perform chemical work is what is left after the physical work is done. Calculations based on the heats of formation alone show frequently such discrepancies between theory and practice that they have fallen into discredit, and it is now recognized that a consideration of the free energies is the true basis for deductions.

Zinc oxide at —273° C. has a heat of formation and therefore a formation energy of 85,800 calories. If it maintained that formation energy at higher temperatures it could not be reduced either by carbon monoxide, or by carbon with the formation of dioxide. In the reduction by carbon with the formation of monoxide a temperature of about 6000° C. would be requisite. Boudouard found, however, that zinc oxide though not reduced by carbon at 1100° C. is perceptibly reduced at 1125 C. At 1125°C., therefore, the formation energy of zinc oxide and that of carbon monoxide must be equal, the value by computation being 42,740 calories. At ordinary temperature the formation energy of zinc oxide is 76,850 calories.

Zinc oxide can only be reduced by carbon monoxide so long as the concentration of the carbon dioxide is very small. Thus carbon monoxide would act reducingly at 1000° C. until the carbon dioxide tenor of the gas amounted to 0·1%. With higher tenor of CO_2 zinc would be oxidized. At 1125° C. the CO_2 may amount to 0·2% and at 1500° C. to 0·76%. The conversion of carbon into monoxide has the most important part in the reduction, as technical experience shows. Besides it, carbon dioxide appears in amounts under 1%. Because of the reduction of the dioxide by carbon

[1] Zts. f. Elektrochemie, VIII, xliv, 833 to 843.

the proportion is never attained, at the high temperatures, which would be capable of existing with the zinc. This is to say that zinc oxide is reducible by carbon with the formation of carbon monoxide at the temperature of 1125° C., as determined by Boudouard, and that the carbon monoxide may further reduce zinc oxide if the carbon dioxide which arises from that reaction be reduced again by carbon to the degree that the percentage of the dioxide does not exceed a certain limit, depending on the temperature.

Necessity for Solid Carbon in Effecting Reduction of Zinc Oxide.—Irrespective of theoretical considerations, both experience and experiment demonstrate that not only is solid carbon necessary in the retort, but also an intimate mixture of it with the ore is requisite, and the more perfect the mixture the better the result. A series of experiments to determine this point was once made at the Wilhelminehütte (Upper Silesia), with very instructive results. In these the ore to be reduced and the reducing agent were charged into a muffle in different layers, separated by pieces of lime. The zinc oxide and carbon were not, therefore, in contact. The yield of zinc obtained from the muffles thus charged was so small that all could be attributed to the reducing action of carbureted hydrogen in the carbonaceous material. Concerning these experiments the following was written:

"On the other hand, the remaining possibility that it is chiefly the solid carbon which reduces the zinc oxide is itself a great riddle. Solid bodies can act chemically on one another only by direct contact, and it is inconceivable that the zinc reduction process can be carried out in the retort in that manner so completely as it is really done. We can not imagine how every atom of zinc oxide in a piece of zinc ore can come in contact with an atom of solid carbon. Whether the active molecular movement of the glowing fragments of ore and carbon play a part in this, or whether at the temperature prevailing in the retort there is formed a lower oxide of carbon (a compound of carbon and carbon monoxide) must remain undetermined. It has been proved, however, by many experiments and the results of actual practice that as intimate a mixture of ore and carbon as possible is desirable; if the ore be too coarse or if the charge be not well mixed the yield of zinc falls off. The ore and reducing material must not be too fine, however, else the charge in the retort will be too dense and the gases developed will not have the necessary freedom to escape."[1]

Theoretical Consumption of Coal in Zinc Distillation.—If the reduction of zinc oxide takes place according to the reaction

$$ZnO+C=Zn+CO,$$

[1] Denkschrift zur Feier des fünfzigjährigen Bestehens der Bergwerksgesellschaft Georg von Giesche's Erben gehörenden Wilhelmine-Zinkhütte zu Schoppinitz, p. 25.

which requires 57,000 heat units (gram-calories), for the production of 65 kg. of zinc there are required theoretically only 57,000 calories, or 877 per kg., corresponding to 0·109 kg. of pure carbon and 0·125 kg. of coal of 7000 calories heat value. This refers only to the heating coal from which the extraneous energy required to effect the reaction must be derived. If the highest grade of ore that can be produced commercially (assaying 75% Zn after roasting) were to be distilled with a consumption of one ton of heating coal per ton of ore and a yield of 90% of its zinc, which are approximately the maximum results that can be expected from the present best practice, the consumption of coal would be about 1·5 kg. per kilogram of spelter produced, or about 12 times the theoretical quantity. Practically, the consumption of coal is greatly in excess of that ratio. Theoretically, 1 kg. of roasted ore containing 75% Zn should require only 0·138 kg. of carbon as reduction material; in practice about 0·4 kg. would be used.

The above estimate is presented merely as an illustration of the great difference between the actual consumption of heating coal and that which is theoretically required. In reality more heat is called for theoretically than what is needed to effect the simple reduction of zinc oxide. Thus lead oxide and ferric oxide present in the charge must also be reduced and some other endothermic reactions must be effected; the inert gangue, coal ash and excess of coal must be raised to the temperature of the furnace, absorbing a certain quantity of heat, which is lost when the incandescent residuum is drawn out of the retort; moisture and volatile matter in the coal must be expelled; and a further quantity of heat is absorbed and rendered latent in the zinc vapor. In order to compute the actual theoretical requirement of heat it would be necessary to take account of all the above items, but even then there would be a large difference between it and that which is furnished by the combustion of the heating coal in the best practice.

What then becomes of the difference? The most part of it is lost in the products of combustion escaping to the chimney, which must leave the furnace at nearly the maximum temperature attained in the latter; this loss is reduced in so far as heat is recuperated from the waste gases, but it is large even under the most favorable conditions. The next largest item of loss is suffered through radiation from the furnace, the large surface of the latter and its necessarily thin front walls tending to promote the dispersion of heat into the surrounding atmosphere; this loss can be reduced only by increasing the thickness of the end walls and especially that of the roof arch so long as the present form of furnaces must be adhered to.

Composition of Retort Gases.—The facts, then, which are known definitely as to the reactions in the retort are: (1) an intimate mixture of the

ore and coal is necessary; (2) an atmosphere of carbon monoxide must be preserved in the retorts; and (3) the gases escaping from the retorts during distillation actually consist chiefly of carbon monoxide. The last is illustrated by analyses of the retort gases at Letmathe, Westphalia, as follows:[1]

Constituents.	I	II	III	IV	V	VI
Carbon dioxide.....	15.58%	0.48%	1.06%	0.11%	1.10%	0.82%
Carbon monoxide....	38.52	undet.	92.16	97.12	undet.	98.04
Hydrogen.....	41.70	5.32	1.83	undet.	undet.	0.72
Nitrogen.....	Trace	undet.	1.01	0.41	undet.	Trace
Methane.....	4.17	.....	.....	.....	.....	.....

The first sample was taken before the beginning of the distillation and the last just before the end. The methane and in part also the hydrogen of the first sample resulted from the gasification of the lean stone coal used as reducing agent; another part of the hydrogen resulted from the decomposition by the glowing coal of the aqueous vapor from the ore. From this decomposition comes also a part of the carbonic monoxide and dioxide, while a part comes, moreover, from the oxidation of the coal by air remaining in the retort after the latter is charged. Analyses of the retort gases during distillation at the Münsterbusch works, Stolberg, showed:

	CO_2	CO	H	N
I.	0·09%	95·36%	3·72%	0.61%
II.	0·11	97·42	1·20	0·92

Ad. Firket reports the following analyses of the gas issuing from the condensers at a Belgian works:

	15 min. after charging.	Immediately before putting on the prolongs.	3 h. after putting on the prolongs.	6 h. after putting on the prolongs
CO_2.....	13·00%	25.00%	2.00%	0.70%
CO.....	nil	13.00	48.00	90.00
SO_2.....	0·15	0.02	nil	nil
O.....	8·00	1.00	2.50	9.00
N.....				
H.....				
CH_4.....	78·85	60.98	47.50	0.30
C_2H_4.....				
H_2O.....				

The percentage of CO continues to increase up to 97 or 98%, while that of CO_2 becomes insignificant and oxygen ceases to appear.[2]

[1] Dingler's Polytech. Journ., CCXXXVII, 390; Berg- u. Hüttenm. Ztg., 1880, p. 371; Wagner's Jahresbericht, 1880, p. 186.

[2] Annales des Mines de Belgique, VI, i, 49.

Formation of Blue Powder.—Unfortunately, all of the zinc vapor issuing from the retort is not condensed as liquid, a part of it passing directly to the solid state in the form of a very finely divided, bluish powder. The proportion of the latter, in good practice, varies from 5 to 10% of all the spelter produced. The tendency toward the formation of blue powder is greatest when the condensers are comparatively cold and the gas is dilute; hence it forms especially at the beginning of the distillation. When prolongs are used on the condensers an additional quantity of zinc is recovered, entirely in the form of blue powder, inasmuch as the cooling of the vapor in them is very rapid and to a relatively very low temperature. The blue powder is a troublesome between-product or by-product, according as to whether it must be resmelted or can be marketed after a further preparation.

There is invariably a considerable percentage of zinc oxide mixed with the blue powder, which is due more or less to oxidation of zinc by aqueous vapor or carbon dioxide. This happens especially at the beginning of the distillation. According to Lencauchez's investigations, zinc is oxidized by CO_2 at a comparatively low temperature and more rapidly with increase in temperature. The more slowly the distillation is performed and the more gradual the cooling of the zinc vapor, the greater is the formation of zinc oxide; rapid distillation and quick condensation of the zinc diminish the percentage of oxide. The oxidizing effect of carbon dioxide on zinc vapor is manifested when gases of combustion find entrance into the retorts through rents in the latter. In general the blue powder collected in distillation contains about 10% ZnO.

Comparative Reducibility of Zinc Oxide Produced in Different Ways.—Zinc oxide produced in various ways differs as to its reducibility by carbon and carbon monoxide. According to von Hauer, zinc oxide precipitated from solutions is reduced more quickly, at a lower temperature and more completely than calcined zinc carbonate ore.[1] Of the natural ores, the calcined carbonates are the most easily reduced. Zinc oxide produced by the roasting of blende requires a higher temperature. It is observed also that the more roasted blende there is in a charge for distillation the more reduction coal is needed, and that there is a difference in the reducibility of zinc oxide obtained from various blendes. The scientific reasons for these phenomena are unknown. There have been no pyrometric investigations of the subject, which presents to chemists and metallurgists an interesting line of work. It may be remarked that a similar variation in the reducibility of iron ores has been

[1] Kerl, Grundriss der Metallhüttenkunde, p. 445.

observed and a review of the investigations as to the causes thereof would be no doubt instructive in studying the behavior of zinc ores.[1] Zinc silicate is more difficultly reduced than zinc oxide obtained by the calcination of either carbonate or blende; its reduction is effected only by solid carbon, whence an intimate admixture with the reduction material is especially important.

Influence of Foreign Matter in the Reduction of Zinc Oxide.—The commonest impurities in zinc ores are silica, alumina, lime, magnesia and the compounds of iron, manganese, lead and cadmium. These elements when heated intensely in the strongly reducing atmosphere of the zinc retort behave in about the same manner as in the lead or copper blast smelting furnace, where the atmosphere is also essentially carbon monoxide, i.e., certain oxides are reduced to lower oxides or metals and the basic oxides combine with the acid to form more or less fusible slags. A study of these conditions is important not only with view to the corrosion of the retorts and danger of loss of zinc therefrom, but also with view to obtaining the maximum yield of zinc from the charge even when the retorts are in no immediate peril. It is only recently that the subject of arranging charge mixtures so as to neutralize the dangerous effects of certain impurities has received any scientific attention and but little experimental work has yet been devoted to it, although it is obviously a promising line of investigation, especially in view of the complex character of the ores that are now offered and constitute important supplies of raw material.

Sulphur.—Sulphur in the ore decreases the yield of zinc, especially if it be combined with the zinc itself, since each unit of sulphur present as zinc sulphide holds back two units of zinc,[2] the sulphide of zinc not being reducible to metal by carbon, or if so only at so slow a rate as not to be technically relied upon at all. Zinc sulphate is chiefly reduced in the retort to zinc sulphide, with development of carbon monoxide ($ZnSO_4+4C=ZnS+4CO$), although when heated with carbon to only red heat zinc sulphate is converted to oxide with the development of sulphurous anhydride and carbonic dioxide. Zinc sulphide at the temperature prevailing in the retort is decomposed by iron ($ZnS+Fe=Zn+FeS$), but the ferrous sulphide is highly corrosive to the retorts and consequently is objectionable. Metallic iron may be formed in the retort by the reduction of ferrous oxide, but the temperature is not ordinarily sufficiently high to produce much iron except

[1] Vide Sir I. Lowthian Bell. Chemical Phenomena of Iron Smelting.

[2] This is the old rule, and a safe one to go by, although it holds true only in the case of pure zinc sulphide, inasmuch as it has been proved that certain of the impurities which commonly occur in zinc ore have an important decomposing effect on ZnS, as is shown further on.

in the case of ores which have a comparatively high tenor in that element.[1] According to Professor Eugène Prost, of the University of Liège, Belgium, the decomposition of zinc sulphide by iron depends at any given temperature upon the proportion of iron, which should always be in excess of the amount theoretically requisite for combination with the sulphur.[2] With a mixture of ZnS and 190% excess of iron the zinc was wholly volatilized at 1200° to 1250° C., while at 1075° considerable remained in the residue. Zinc sulphide is also decomposed by lead oxide, with the formation of zinc oxide, lead and sulphurous anhydride.[3] The reaction of zinc sulphide and zinc oxide is uncertain. Percy concluded that they formed zinc and sulphurous anhydride, analogous to the reaction between copper sulphide and copper oxide, while Berthier stated that they combine to form fusible oxysulphides. Subsequent experiments have shown that the reaction reported by Percy does take place, but is incomplete and is useless as the basis of an industrial process. Zinc sulphide is decomposed by lime (only in the presence of carbon, according to Berthier [4]) with the formation of zinc and calcium sulphide. Prost found by experiment [5] that lime was a more efficient desulphurizing agent than iron, an excess of 75% over the theoretical quantity decomposing zinc sulphide completely at 1250°, while with a larger proportion of lime the same result could be obtained at 1200° C. Prost's experiments were made with a pure, crystalline sphalerite.

Calcareous zinc ores usually show a comparatively high tenor in sulphur after roasting, owing to the formation of calcium sulphate. According to Steger, roasted blende from the Neuhof mine, Upper Silesia, showed 8 to 9% S as sulphates, chiefly combined with lime and magnesia, and only 0·03 to 0·8% as sulphide, chiefly combined with zinc.[6] Both calcium sulphate and barium sulphate are reduced to sulphides in the retort, and it used to be thought that those substances led to loss of zinc by the formation of zinc sulphide according to the reaction [7]

[1] In Belgium and Westphalia metallic iron is sometimes recovered from the retort residues. At Letmathe, Westphalia, where ores high in iron, including some calamine with 30% Fe, were distilled, there was recovered monthly 30 tons of iron, high in sulphur (Th. Dahlblom, Berg- u. Hüttenm. Ztg., Dec. 11, 1891, p. 450). In the distillation of a ferruginous Colorado blende in Sadtler retorts, masses of metallic iron weighing several pounds formed in the retorts. E. Jensch, in Chem. Ztg., 1894, pp. 101-102, described a mass of iron weighing 1 kg. which was formed in a zinc retort, wherein the temperature had not been in excess of 1500° C. It contained 84 to 87% Fe, 1 to 2% Si, 1 to 2% C, and several units of sulphur, the composition not being uniform throughout the mass. It was therefore essentially a mixture of pig iron with iron sulphide and oxide.

[2] Bull. de l'Assoc. Belge des Chimistes, X, vi, 246 to 263; Journ. Soc. Chem. Ind., 1896, p. 813.

[3] Berthier, Tr. d. Essais, I, 403.

[4] Op. cit, II, 570.

[5] Loc. cit.

[6] Zts. f. d. Berg- Hütten- u. Salinenwesen in Preuss. Staate, L, iii, 507.

[7] Thum, Berg- u. Hüttenm. Ztg., 1876, p. 154; Schnabel, Handbuch der Metallhüttenkunde, II, 94.

$$2CaO,\ SO_3+Zn+4C=Ca_2S+ZnS+4CO_2,$$

or possibly [1]

$$CaS+ZnO=ZnS+CaO.$$

Barium sulphate was supposed to play the same part as calcium sulphate. Recently Prost has shown [2] experimentally that at 1200° to 1250° C. calcium sulphate exerts no retarding influence on the reduction of zinc oxide, not even in presence of lead and iron silicates, which might convert reduced calcium sulphide to silicate and facilitate the formation of zinc sulphide. It is fair to presume that barium sulphate is equally harmless.[3] On the other hand magnesium sulphate, which at high temperature is decomposed by carbon into magnesia and sulphurous anhydride, favors the production of zinc sulphide.

E. Landsberg proposed, 20 years ago, to roast blende partially and then to reduce in the retort with the admixture of lime, the following reaction being supposed to take place:

$$ZnO+ZnS+CaO+C=2Zn+CaS+CO_2,$$

but the idea does not appear to have had any practical application.[4] According to H. M. Taquet [5] zinc sulphate is reduced by calcium oxide and carbon, calcium sulphide being formed according to the following reaction:

$$ZnSO_4+CaO+5C=Zn+CaS+5CO.$$

Professor Prost, whose experimental work in zinc distillation has been of the highest importance, recently made some interesting investigations at a Belgian smeltery on the influence of the degree of desulphurization of blende upon the yield of zinc in distillation.[6] Two ores of essentially different character, one plumbiferous, the other ferruginous, were selected for investigation. Their compositions before and after roasting were as follows:

	Zn	Pb	Fe	CaO	MgO	SiO_2	S
Plumbiferous.....							
Before roasting..	49·0%	10·0%	1·1%	0·5%	0·4%	1·2%	19·0%
After roasting..	60·0	11·0	1·5	0·6	0·5	1·5	{ 2·00 2·25
Ferruginous.....							
Before roasting..	36·0	4·0	15·0	0·9	0·6	5·0	30·0
After roasting..	42·5	4·7	18·0	1·1	0·7	6·0	2·9

[1] Kerl, Grundriss der Metallhüttenkunde, p. 431.

[2] Loc. cit.

[3] For a fuller discussion of this subject, refer to p. 33.

[4] Berg- u. Hüttenm. Ztg., 1879, p. 104; Chem. Ztg., 1879, p. 3; German patent, No. 23,280.

[5] German patent, No. 137,004, March 21, 1901.

[6] Rev. Univers. des Mines, XLIX, iii, p. 247, March, 1900.

Experiments were made with the ore completely roasted and incompletely roasted. The roasted ores gave the following results upon chemical analysis:

	Plumbiferous ore roasted.		Ferruginous ore roasted.	
	Completely.	Incompletely.	Completely.	Incompletely.
Total sulphur	2·10%	5·33%	2·91%	7·73%
S as sulphates sol. in H_2O	0·43	0·50	1·28	0·87
Zn as $ZnSO_4$, sol. in H_2O	0·45	0·32	1·12	0·63
Ca " $CaSO_4$, " " "	0·52	0·63	1·14	1·06
Mg " $MgSO_4$, " " "	0·10	0·12	0·27	0·18
S as $PbSO_4$, sol. in ammon. tartrate	1·34	0·31	0·72	0·36
S existing in other forms	0·33	4·52	0·91	6·50

In the case of the ferruginous ore the rise in the percentages of sulphates at the end of the roasting is noteworthy. This was attributed to the formation of SO_3 at the expense of the undecomposed sulphides in the last stage of the process.

The reduction tests were made with 125 kg. of each kind of ore. The dead roasted samples were mixed with 85 kg. of lean coal; the partially roasted with 75 kg. The samples of each pair were charged in alternate retorts of the same row in the furnace. They were therefore subjected to identical conditions. The distillation lasted 24 hours, the retorts being handled along with those of the furnace which contained the regular works charge, but the residues after distillation were carefully kept separate. The results were as follows:

Sample.	Kg. ore charged.	Kg. coal charged.	Weight of residues kg.	Zinc in ore charged.	Zinc in residues.	Iron in residues.	Sulphur in residues.	Sulphur in ore charged.	Percentage of total zinc retained by residues.
I.	125	85	68·50	59·28%	1·46%	5·06%	3·70%	2·10%	1·35%
II.	125	75	78·70	58·97	6·92	5·90	8·02	5·43	7·38
III.	125	85	73·35	44·87	0·73	24·20	4·07	2·91	0·95
IV.	125	75	92·60	43·10	3·90	19·10	8·26	7·73	6·70

I. Plumbiferous blende, completely roasted; II. do., incompletely roasted; III. ferruginous blende, completely roasted; IV. do., incompletely roasted.

The above results demonstrate that too much care cannot be exercised in effecting the desulphurization of blende. However, in comparing the results obtained with the partially roasted plumbiferous ore and the partially roasted ferruginous ore, taking into account the difference in their tenors of

sulphur, it appears that the loss is less important in the case of the ferruginous mineral than in that of the plumbiferous. This is due without doubt to the decomposition of zinc sulphide by iron. The tenor of zinc, namely 3·9%, in the residue from the distillation charge of incompletely roasted ferruginous ore is high compared with the tenor of the residue from the completely roasted ore, but nevertheless it does not exceed the tenor of zinc in the ashes of mixed ores which are regularly smelted at Belgian works. Prost raises the question whether the zinc sulphide existing in the retort residues is not due to physical causes rather than to chemical, pointing out that although zinc sulphide is difficultly reducible by carbon, nevertheless it is reduced at the temperature prevailing in the distillation furnace, and although the reaction may be very slow it must be remembered that the ore remains 24 hours in the retorts and during that time is subjected to an extremely elevated temperature. He ventures the theory that the incompletely roasted ore contained relatively large particles of undecomposed sulphide, which because of insufficiently intimate contact with the carbon in the retort escaped its reducing action, whereas if the ore had remained longer in the roasting furnace, these large particles would have perhaps been decomposed.

Iron and Manganese.—Iron occurs usually in calcined and roasted zinc ores in the form of ferric oxide. As sulphide it ought never to be present, since iron sulphide is oxidized more easily than zinc sulphide and the total sulphur in the roasted ore being probably less than 1% this would naturally be assumed to be chiefly in combination with zinc, although Jensch stated[1] that he found that the sulphur remaining in a roasted ferruginous blende was present as iron sulphide. Iron monosulphide, if present in the retort, would simply be melted, in which form it would have a highly corrosive action on the retort, like the same substance originating from the decomposition of zinc sulphide by iron. Ferric oxide is reduced to ferrous oxide, which combines with silica or acid silicates in the ore and other bases, e.g., lime, forming fusible slags. Where these basic slags come in contact with the walls of the retort, they combine with the aluminum silicate and free silica and gradually eat through them; destruction of retorts signifies not only loss of their value, but also loss of zinc. The formation of fusible slags is objectionable, moreover, because they envelop particles of zinc oxide which is thus removed from direct contact with the reducing agent and consequently escapes reduction. A similar loss may result from the actual scorification of zinc oxide as silicate in very ferruginous slags, such slags having the capacity of taking as much as 20% zinc oxide into their composition and still remaining fluid; the more lime and

[1] Berg- u. Hüttenm. Ztg., 1894, p. 299.

the less iron there is in the slag, however, the less is the tendency of zinc to enter it. Zinc silicate is reducible by carbon, but direct contact is necessary, and its formation in the retort charge would be apt to separate it from the carbon, especially if the original mixture were not as intimate as it ought to have been. Iron is consequently a highly injurious impurity in a zinc ore and it is desirable always to reduce the percentage of it so far as practicable in the preliminary dressing. Manganese behaves in a precisely similar manner, but the manganiferous slags are if anything more liquid and corrosive than those which are only ferruginous. With good retorts and a judicious compounding of the charge a tenor of 10% Fe+Mn may be permitted without fear of disastrous results. Some Belgian smelters regularly work charges with that percentage, and at certain places ores even higher in iron are reduced. In the United States, the New Jersey and Pennsylvania smelters used to treat an ore with about 10% Fe+Mn, but the Western smelters have never tolerated anything like that.

Of all the impurities that occur in zinc ores, sulphur combined with iron, or in such form that a combination with iron may effect itself in the retort, is probably the worst. "When the ore is very ferruginous and at the same time contains 2 to 3% S, it may be expected that the smelting will give bad results and a large consumption of retorts."[1] The important reaction by which ferrous sulphide reduces the infusible sesquioxide of iron to the slag forming protoxide is shown by the following equation:

$$FeS+3Fe_2O_3+7SiO_2=7FeO,SiO_2+SO_2.$$

Lead.—Lead may be present in the ore as sulphide, sulphate, oxide and silicate. If the ore be a roasted sulphide the major part of the lead will be in the form of sulphate (vide p. 32 and p. 209) and silicate. At the temperature which exists in the retort, lead sulphate may undergo numerous reactions. It may be decomposed by silica with the formation of lead silicate and sulphurous anhydride. It may be reduced by carbon to sulphide and the latter may react with another portion of sulphate to form metallic lead, or it may be decomposed by either iron or lime with the formation of metallic lead. Lead silicate is reduced partially by carbon and is decomposed by iron and ferrous oxide and to some extent by ferrous sulphide. Free lead silicate acts corrosively on the retorts, as do also the fluid slags formed through its decomposition, which have the drawback also of enveloping particles of zinc oxide and thus preventing their reduction. Lead oxide is easily reduced by carbon. The lead reduced in these various ways is partly volatilized and condensed with the zinc, contaminating the latter,

[1] De Sinçay, The Mineral Industry, VIII, 657.

and partly remains in the retort residues, whence it may be recovered as metallic shot or prills; the amount of lead condensed with the zinc is not apt to exceed 3% of the weight of the latter, and the more part of the lead in the ore remains behind in the retort residue, from which it may be partially recovered by jigging.

Lead was formerly thought to be a more objectionable element in the retort than it is nowadays. One metallurgist wrote in 1880: "The influence of lead in any state is bad, the silicate formed thereby being very corrosive, while a large proportion of the metal passes into the spelter of which the quality is injured. More than 2% lead should not be tolerated in a calamine and a penalty of $1 per unit per ton of ore should be inflicted for any excess over that limit." Experience has proved, however, that with modern, well-made retorts there is not much danger to the latter from lead, and the degree of contamination of the spelter can be to a large extent controlled.

At the Wilhelminehütte (at Schoppinitz), Upper Silesia, one side of a furnace was charged regularly with calcined calamine, which averaged 10% Pb (originally in the form of cerussite), but occasionally contained as much as 25%, while the other side of the furnace was run on ordinary ore. The result did not show any increase in the breakage of the retorts working on the lead-bearing, and afterward ore of that kind was regularly treated at the Paulshütte, the lead-holding ashes being subsequently smelted in shaft furnaces at the Walter-Croneckhütte at Rosdzin.[1] The Silesian zinc ores generally contain lead although not a high percentage thereof, and the conditions of the smelting process practised in that district result in a rather impure spelter, which has commonly to be subsequently refined.

Cadmium.—Cadmium oxide is reduced to metal, which is volatilized chiefly during the early stages of the distillation. A part of the cadmium that goes over is condensed with the zinc, but owing to its greater volatility the more part escapes from the condenser as fume and is reoxidized in the atmosphere. In the Silesian process of distillation this oxide may be collected to serve as a source for the recovery of the metal.

Gold and Silver.—Gold and silver contained in zinc ores are not volatilized to any important extent, but remain in the retort residues, whence they may be recovered. If the ore contains lead compounds the metal reduced from the latter will take up silver and gold with which it may come in contact, just as in the lead smelting furnace, and by jigging the residues the prills of argentiferous lead may be obtained. Otherwise the silver and gold may be recovered by smelting all the residuum together with lead ores

[1] Denkschrift zur Feier des Fünfzigjahrigen Bestehens der Bergwerksgesellschaft Georg von Giesche's Erben gehörenden Wilhelmine-Zinkhütte zu Schoppinitz, p. 28.

in the ordinary manner, but the tenor in precious metals would have to be considerable to make this a profitable process.

Advantage is taken of the non-volatility of silver and gold to separate by distillation the zinc from the crusts obtained in the Parkes process of lead desilverization. The zinc-gold precipitate obtained in the bromocyanide process of gold extraction at Deloro, Canada, is also treated in the same manner, which is equally applicable to the precipitate got in the ordinary cyanide process, and under certain circumstances might be the cheapest method of dealing with the latter. At Deloro, precipitate assaying 20% gold is distilled and the zinc and zinc oxide condensed contains about 0·1% of the gold charged, that amount being probably carried over mechanically.

Fluorspar.—Fluorspar is an undesirable constituent of zinc ore because it forms easily fusible slags, which attack the retorts.[1] The behavior of fluorspar in blende roasting is discussed on p. 34.

Arsenic and Antimony.—Arsenical and antimonial compounds are partially reduced and partially volatilized; if present in the ore traces of arsenic and antimony will appear in the spelter.

Silica, Alumina and Lime.—Quartz behaves in the zinc retort precisely as in lead and copper smelting furnaces, combining with bases to form more or less fusible slags. Quartz alone is inert. Alumina may combine with quartz, forming a silicate, or with zinc oxide, forming an aluminate, or zinc spinel. This may be observed often in the walls of old retorts which have been penetrated by zinc vapor and oxidized. Lime alone has a beneficent influence, tending to decompose zinc sulphide. Lime and quartz together are not particularly harmful, since calcium silicate is not easily fusible, but when iron or manganese oxides are present liquid slags will be formed. Magnesia behaves like lime, but magnesian slags are less fusible than the calcareous.

General Principles Governing the Compounding of Charges for Distillation.—The chemical reactions which take place in the retort are therefore analogous to those of a blast smelting furnace, or more precisely to what would take place in a crucible in a pot-furnace, an atmosphere of carbon monoxide being maintained in the crucible. Originally the retort is charged nearly full with a mixture of fine ore and coal, the charge comprising, say, 25 kg. of ore and 10 kg. of coal; if the ore assays 50% Zn there is about 15·6 kg. of ZnO in the charged retort. During the distillation perhaps 95% of the zinc may be reduced and volatilized and 50% of the coal may be burned, so that the contents of the retort are constantly diminished and at the end may consist, say, of 1 kg. of zinc oxide and sulphide, 10 kg. of

[1] Foehr, Chem. Ztg., 1890; Mineral Resources of the United States, 1889-1890.

gangue and ash of coal burned, and 5 kg. of unburned coal, a total of 16 kg. The contents of the retort have, consequently, been decreased about 54%. As the zinc is eliminated and the coal burned out, the particles of foreign mineral in the ore come more into contact and unite in a slag or cinder if they have affinity to each other. If the slag be fusible it trickles down through the residue to the bottom of the retort on which it runs forward because of the inclination of the latter toward the front of the furnace. If the slag be basic it takes up silica from the retort, which is thereby corroded. The flow of the slag is, of course, impeded by the fine particles of coke around which it may coalesce.

It follows, therefore, that comparatively little danger exists so long as the charge does not contain components which will form a fluid slag. Even if the ore does not contain free silica or acid silicates there is always silica present in the ash of the coke. Lime or magnesia are not to be dreaded alone, because their silicates are difficultly fusible. Even iron or manganese alone would not be very dangerous with a calamine ore or a thoroughly well roasted sulphide. But silica, iron and lime together may cause trouble.

In the formation of singulo-silicates one part of silica by weight combines with 1·86 of lime, 1·33 of magnesia, 1·14 of alumina, 2·40 of ferrous oxide 2·36 of manganous oxide, and 7·43 of lead oxide; for bisilicates the proportion of the base is one half that required for singulo-silicates, and for trisilicates one third. Conversely, one part of lime requires 0·535 of silica for a singulo-silicate, one part of magnesia requires 0·750, one of alumina 0·873, one of ferrous oxide 0·416, one of manganous oxide 0·422, and one of lead oxide 0·134. For bisilicates the proportion of silica is doubled, and for trisilicates it is tripled.

In mixing zinc ores of various compositions to make a charge for distillation, the ores should be combined in such a way as to produce either a purely silicious or a purely basic mixture. If that be infeasible, the mixture should be designed so that silica will form with the bases a slag as difficultly fusible as possible at the temperature prevailing in the retorts, which at its maximum is about 200° C. above the temperature possessed by the slags issuing from a lead blast furnace. In order to make difficultly fusible slags large quantities of lime, or better lime and magnesia, or alumina should be combined with only a small quantity of silica; or on the other hand if the ore be silicious, only a small quantity of earthy bases should be added. It should also be borne in mind that a mixture of silicates fuses generally at a temperature below the calculated average of the several silicates alone. Thus aluminum silicate is by itself fusible only at

an extremely high temperature, but in the presence of lime, iron or manganese it will combine with those bases to form a slag which will melt at a low temperature. The most fusible silicates are formed in the presence of manganese, iron, soda and potash and the percentage of those elements in the ore to be distilled should be kept low by admixture of other ores which are free from them.

VII.

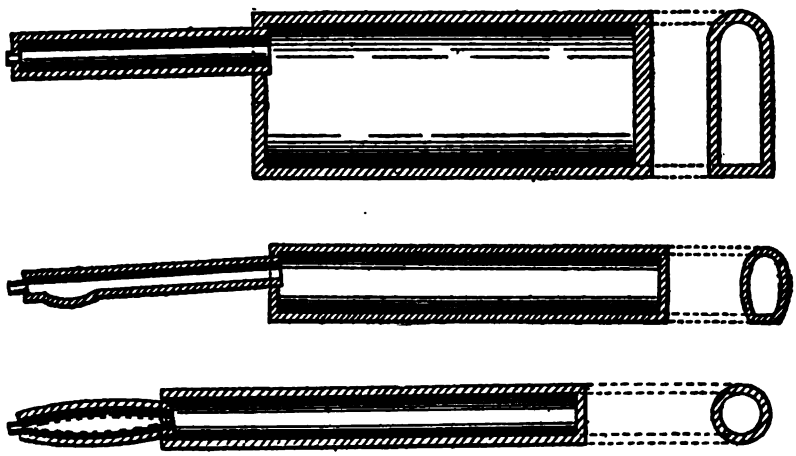
RETORT AND CONDENSER MANUFACTURE.

Before taking up the further treatment of the roasted ore, it is proper to consider another preliminary step, namely retort and condenser manufacture, which is of prime importance in the subsequent process. Upon this as much as anything else indeed does the success of zinc distillation depend, since large losses may occur through bad retorts not only in the residues (because poor retorts will not withstand the requisite heat) but also by absorption in the clay of the retort and by escape of vapor through the pores of the latter into the combustion chamber of the furnace, and thence up the chimney; and it is in this direction, i.e., improvement of the retorts, that there has been one of the most important advances in zinc winning in Europe during the last 20 years.

TYPES OF RETORTS.—The distillation vessels used in all types of furnaces are retorts, properly speaking. The peculiar retorts which are employed in Upper Silesia are known as "muffles" (Muffeln), on account of their form; the vessels employed in the Belgian furnaces are called by the Germans "Röhren" (tubes), by the French "creusets" (crucibles), and by the Welsh "pots"; in the United States, where only one kind of retort is used, namely the small cylindrical or elliptical vessel, there is no distinctive word, and they are referred to simply as retorts. In this treatise I shall follow the American custom and by the word retort refer to the distillation vessel in general and the small cylindrical tube, crucible or pot in especial, referring where necessary to the large vessel used in Upper Silesia as muffles. This classification will cover the medium size vessels used in Rhenish Prussia and Westphalia, which partake of the nature of both crucibles and muffles.

The retorts employed in zinc distillation are clay vessels of three general types: (1) The large muffle-shaped vessel used in Upper Silesia; (2) a medium size used in Rhenish Prussia and Westphalia and to a slight extent in Upper Silesia; and (3) the relatively small vessel which is employed distinctively with the so-called Belgian process. The three different

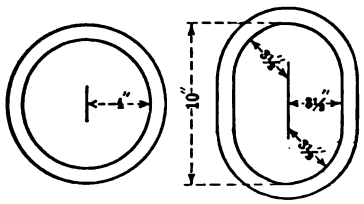
types are illustrated in the accompanying engravings. In the Silesian and Rhenish types of retorts there is little variation among various works; in the Belgian type, on the other hand, there are considerable differences in practice; but Belgian retorts were made originally of circular cross-section, and that is still the commonest form in the United States. In Belgium, however, the standard form at the present day has an elliptical cross-section, or more correctly speaking, the shape of a circle divided diametrically and the two parts separated by a rectangle, the resulting figure being therefore more or less of the shape of the link of a chain. In Wales there are, or were, works which employ retorts of oval cross-section.



FIGS. 157 TO 159. DIFFERENT FORMS OF RETORTS AND CONDENSERS.
Longitudinal and transverse sections.
Fig. 157: Silesian. Fig. 158: Rhenish. Fig. 159: Belgian.

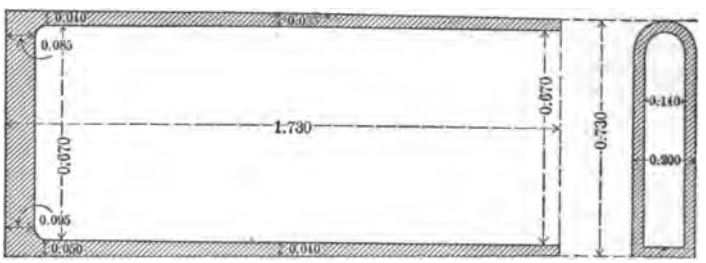
It will be observed from the data given subsequently that in Belgium the elliptical form of retort is now by far the more common, the cylindrical retorts which are still employed being almost entirely in connection with the older furnaces. The advantages of the elliptical form are obvious. Considered as beams supported at each end they are significantly stronger than cylindrical retorts of the same sectional area, which permits them to be made both longer and to be loaded more heavily. They can also be made of larger sectional area, while giving the heat the same facility of penetrating the charge; this is illustrated in the accompanying diagram. Moreover, advantage can be taken of the increased strength of the form to diminish the thickness of the walls, which is, of course, desirable.

DIMENSIONS OF RETORTS.—A common size of the Silesian retort is 0·14 m. wide and 0·60 m. high at the front end, and 1·7 m. long. The height is sometimes made as great as 0·67 m. (Wilhelminehütte), the width up to 0·2 m. and the length 2·15 m., but the last rarely, 1·75 m. seldom being exceeded. The walls of the Silesian retort are made somewhat thicker at



FIGS. 160 AND 161. SECTIONS OF RETORTS.
Diagram showing relative sectional area and heat penetrability of two forms of retorts.

the back than at the front, and the retort itself is sometimes made a little wider there, tapering slightly toward the front. In the thinnest parts, near the front end, the walls are made 25 to 30 mm. thick, increasing toward the rear end to 40 or 50 mm. for top and bottom. A general size for the Rhenish retorts of muffle shape is 0·15 m. wide at the bottom, 0·34 m. high and 1·45 m. long inside (Münsterbusch works at Stolberg); another

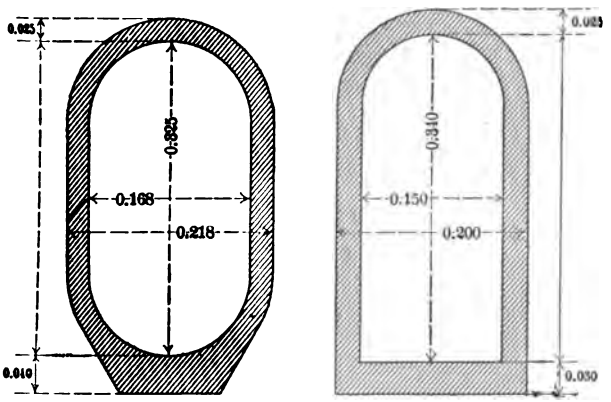


FIGS. 162 AND 163. LONGITUDINAL AND TRANSVERSE SECTIONS OF SILESIAN MUFFLE.
Dimensions in meters.

common Rhenish type is of link cross-section 0·168 m. wide and 0·325 m. high, the walls having a uniform thickness of 0·025 m. except at the bottom, where the thickness is 0·04 m., as shown in Fig. 164. The latter form is the more modern and now the more generally used. In general the Belgian cylindrical retorts have an internal diameter of 8 in. and an external length of 48 to 54 in.; the elliptical retorts of this type are made

to have about the same internal volume as the cylindrical. The extreme ranges of the dimensions of elliptical retorts used in Belgium are 0·195 to 0·300 m. for the major axis and 0·16 to 0·18 for the minor (inside measurements), while the length varies from 1·25 to 1·5 m., very seldom reaching the latter. In the United States the thickness of the walls is generally about 1·25 in. or more; in Europe the manufacture of stronger and denser retorts has made it possible to reduce the thickness of the walls, which is advantageous, and 30 mm. is now rarely exceeded there, while retorts with walls only 25 mm. and even so thin as 22 mm. are commonly to be found.

The length of retorts is restricted by their own weakness and by the



FIGS. 164 AND 165. TYPES OF RHENISH RETORTS.
Transverse sections. Dimensions in meters.

ability of the workmen to charge them. With respect to the latter limitation it is impossible to charge properly a large Silesian muffle in the necessary manner; that is to say, with a long narrow scoop by which the ore must be thrown through the usual condenser, if the muffle be more than 2 m. in length.[1] The length of the Belgian and Rhenish retorts, which are supported in the furnace only at the two ends, is limited chiefly by their lack of transverse strength. Experience has demonstrated that a length of 1·45 m. (57·09 in.) for the Belgian retort and 1·75 m. for the Silesian are approximately the maxima possible with the respective types of furnace.[2]

[1] In Upper Silesia the retorts are charged through the condensers, which are not removed for the maneuver.

[2] Thum laid down the rule that Belgian retorts should not be more than 1·2 m. in length and 16 to 17 cm. in diameter, the thickness of the walls being 25 to 30 mm. at the mouth and 40 to 42 mm. toward the

The dimensions of retorts used in zinc smelting by the various processes both in Europe and the United States, are tabulated in the subjoined statement, in which the dimensions given in roman figures represent meters and those in italics are inches:

DIMENSIONS OF RETORTS USED IN BELGIUM.

Place.	Section	Length	Diameter
Valentin-Cocq.....	Elliptical	1·40 *55·12*	0·254 × 0·168 *10·16 × 6·75*
Flône.....	"	1·34 *53·20*	0·230 × 0·160 *9·20 × 6·40*
Angleur.....	"	1·40 *55·12*	0·230 × 0·160 *9·20 × 6·40*
Ougrée.....	"	1·33 to 1·52 *52·36 to 59·84*	0·200 × 0·180 *8·00 × 7·20*
Bleyberg.....	Circular	1·17 to 1·37 *46·06 to 53·94*	0·160 *6·40*
"	Elliptical	1·23 to 1·47 *48·43 to 57·87*	0.200 × 0.180 *8·00 × 7·20*
Boom.....	Circular	1·35 *53·15*	0·17 *6·80*
Prayon.....	Elliptical	1·27 to 1·45 *50·00 to 57·09*	0·225 × 0·160 *9·00 × 6·40*
"	"	1·25 to 1·60 *49·21 to 63·00*	0·255 × 0·180 *10·20 × 7·20*
Overpelt.....	"	1·60 *63·00*	0·300 × 0·170 *12·00 × 6·80*
Seilles.....	"	1·45 *57·09*	0·205 × 0·185 *8·20 × 7·40*
Corphalie.....	Circular	1·20 to 1·33 *47·24 to 52·36*	0·200 *8·00*
Engis.....	"	1·25 to 1·30 *49·21 to 51·18*	0·170 *6·80*
".....	Elliptical	1·25 to 1·30 *49·21 to 51·18*	0·195 × 0·175 *7·80 × 7·00*
"	"	1·25 to 1·65 *49·21 to 64·96*	0·260 × 0·180 *9·20 × 7·20*
Antheit.....	"	1·35 *53·15*	0·209 × 0·178 *8·36 × 7·12*

DIMENSIONS OF RETORTS USED IN RHENISH PRUSSIA AND WESTPHALIA.

Place	Section	Length	Diameter
Stolberg.....	Muffle	1·45 *57·09*	0·34 × 0·15 *13·60 × 6·00*
"	Elliptical	1·45 *57·09*	0·325 × 0·168 *13·00 × 6·72*
Hamborn.....	.....	1·40 *55·12*	0·30 × 0·16 *12·00 × 6·40*
Dortmund.....	.....	1·25 *49·21*	0·31 × 0·16 *12.40 × 6·40*

butt (Berg- u. Hüttenm. Ztg., 1878, p. 302). Thum was, however, a strong advocate of the Silesian system of distillation as against the Belgian, the two systems being still quite distinct in his day. Laur had previously vouchsafed the rule that the thickness of the charge should not exceed 18 cm. and the length of the retort 1·5 m., while its height might attain 50 cm. as a maximum (Bull. de la Soc. de l'Ind. Minerale, 1874, III, 395 et seq.).

DIMENSIONS OF RETORTS USED IN UPPER SILESIA.

Place	Section	Length	Diameter
Silesiahütte.....	Muffle	1·73 *68·11*	0·67 × 0·14 *26·80 × 5·60*
Wilhelminehütte.....	"	1·70 *66·93*	0·60 × 0·14 *24·00 × 5·60*
Hohenlohehütte.....	"	1·66 *65·33*	0·56 × 0·15 *22·40 × 6·00*
" a....	"	1·66 *65·33*	0·36 × 0·15 *14·40 × 6·00*

a Experimental; in use in 1893.

DIMENSIONS OF RETORTS USED IN THE UNITED STATES.

Place	Cross-Section	Length	Diameter
Carondelet.....	Circular	....	*8·00*
Collinsville.....	"	*48·00*	*8·00*
Friedensville.....	"	*46·00*	*8·00*
Lasalle.....	"	*54·00*	*8·25*
Jersey City.....	"	*54·00*	*7·00*
" "	Elliptical	*54·00*	*9·00 × 7·00*
Peru.....	"		*11·00 × 7·50*
Pulaski.....	"	....	*10·00 × 8·00*
Pittsburg.....	Circular	*46* to *50*	*8·00*
So. Bethlehem.....	Elliptical	*51·00*	*12·50 × 6·75*

CAPACITY AND WEIGHT OF CYLINDRICAL RETORTS.
(Calculated for walls 1 in. thick and butts 2 in. thick.)

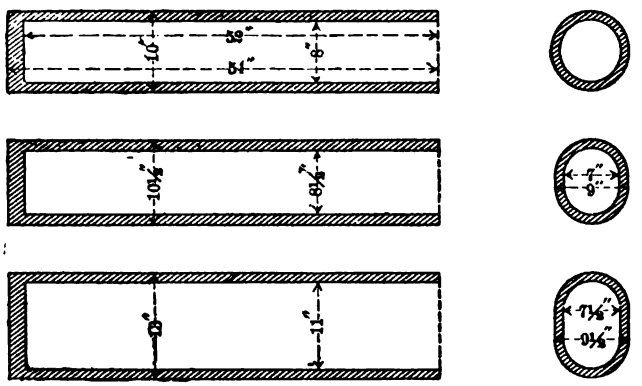
Length outside.		7 in. diam. inside.			8 in. diam. inside.		
Inches.	Meters.	Outside volume cu. in.	Inside volume cu. in.	Weight lbs.a	Outside volume cu. in.	Inside volume cu. in.	Weight lbs.a
46	1·168	2,926	1,693	86·3	3,613	2,212	98·1
47	1·194	2,990	1,732	88·0	3,691	2,262	100·0
48	1·219	3,054	1,770	89·9	3,770	2,312	102·1
49	1·245	3,117	1,809	91·6	3,848	2,362	104·0
50	1·270	3,181	1,847	93·4	3,927	2,413	106·0
51	1·295	3,244	1,886	95·1	4,006	2,463	108·0
52	1·321	3,308	1,924	96·9	4,084	2,513	110·0
53	1·346	3,372	1,963	98·6	4,163	2,564	112·0
54	1·372	3,435	2,001	100·4	4,241	2,614	113·9
55	1·397	3,499	2,040	102·1	4,320	2,664	115·9
56	1·422	3,563	2,078	104·0	4,398	2,714	117·9
57	1·445	3,626	2,117	105·6	4,477	2,764	119·9
58	1·473	3,690	2,155	107·5	4,555	2,813	121·9

a Weight after burning. To find the weight of clay required calculations should be based upon the proportion of chamotte that is used and the percentage of water in the raw clay.
Note: 1728 cu. in. = 1 cu. ft.; 1 cu. m. = 35·314 cu. ft.; 1 cu. decimeter = 61·023 cu. in.; 1 inch = 0·0254 m.; 1 m. = 39·37 in.; 1 kg. = 2·2046 lb.; 1 lb. = 0·45359 kg.

A retort 8 in. in diameter inside and 10·5 in. outside, 50 in. long has an exterior volume of 4330 cu. in. and weighs 134 lb. when burned. The

volume increases 86·6 cu. in. and the weight 2·5 lb. for each inch of length. One of the same dimensions, but 11 in. in external diameter, has a volume of 4752 cu. in. and weight of 164 lb., increasing 95 cu. in. and 3·15 lb. for each inch of length.

Silesian muffles weigh 200 to 260 kg., according to their size and thickness of walls. They take an average charge of 75 to 105 kg. of ore, besides the reduction coal and the between products which are returned. The 0·15×0·35×1·45 m. retorts used in Rhenish Prussia weigh about 70 kg. and take an average charge of about 34 kg. of ore. Belgian retorts are charged variously with from 20 to 27 kg. of ore in the case of the Liège furnaces and 35 to 40 kg. in the case of the Siemens furnaces. These figures refer to the average quantity of fresh ore. For example, 112 retorts



FIGS. 166 TO 168. TYPES OF BELGIAN RETORTS.
Longitudinal and transverse sections.

may be charged with 5000 lb. of ore, or 44 lb. (20 kg.) per retort, as sometimes in Kansas, but in addition there may be 2000 lb. of between products, making the total charge 62·5 lb. (28·5 kg.) of ore, not including reduction material.

REQUIREMENTS OF RETORTS.—There are many requisites in preparing retorts for zinc distillation. They must be of refractory material which will resist intense heat and will not be too quickly corroded by the impurities of the ore. They must be so dense that the zinc vapor will not penetrate them; and they must be strong enough to preserve their shape when loaded with perhaps 60 kg. of ore and coal and heated to a temperature of 1300° to 1400° C. (white heat); the Silesian muffle is, however, supported evenly on its base and therefore does not need to have so great a trans-

verse strength as the Belgian and Rhenish. While fulfilling the above mentioned requirements, the retort must be so thin that the heat of the furnace may act upon its contents as strongly as possible; in other words it must be able to conduct from the exterior to the interior with a sufficient rapidity the quantity of heat which is absorbed by the endothermic reaction taking place inside and at the same time maintain the temperature necessary for the continuance of that reaction. It is because zinc oxide must be reduced in such small vessels and the limitation of the conditions with respect to them, that the process of zinc distillation is so costly and wasteful.

KINDS OF CLAY SUITABLE FOR RETORT MANUFACTURE.—The suitability of various kinds of fire clay for retort manufacture can be determined satisfactorily only by actual trial in the furnace of a specimen set of retorts. A chemical analysis of the material alone is a very unsatisfactory guide. The proper mixture of clay having been once determined, it is so troublesome to make a change for experimental purposes, and the retorts having to be kept for so long a time before use, that the zinc smelter once satisfied

FIG. 169. WELSH-BELGIAN RETORT.
Longitudinal and transverse sections.

with the character of his retorts is disinclined to try anything else. From a manufacturing standpoint the chief requisites for retorts are that they shall be highly refractory and shall burn dense and free from rents. Such material is obtained by mixture of chamotte with just enough clay to bind the chamotte grains, the proportion of clay being kept as low as possible, since in burning the retort the clay shrinks, and favors the formation of cracks. The best clay is chosen for this purpose, but here the difficulty presents itself that the most refractory clays are those which have the greatest shrinkage, while the dense burning clays—the sandy clays—have less durability. This is overcome by a judicious mixture of the two kinds, remembering always that the fusing point of the mixture is lower than the arithmetical mean of the fusing points of the two components.

CHEMICAL AND PHYSICAL REQUIREMENTS.—In general clay for zinc retorts should be low in ferric oxide, lime and alkalies, which increase its fusibility, and should not have much free silica if it is to be used for an ore with a basic gangue; for a silicious ore a more quartzose clay may be

employed. Physically the clay should be of such character that it can be made into a plastic, tough, tenacious paste. The strength of the retort and its impermeability will depend chiefly upon the proportion of clay and chamotte used in the batch, the size of the clay particles and the method of manufacture. Raw clay alone is never used in making zinc retorts, a mixture of raw clay with burnt clay (chamotte) and the crushed material obtained from old retorts or muffles, together with perhaps some other constituents, being generally employed. The composition of the batch varies within wide limits at different works; this is doubtless because each has found by experience that a certain formula gives the best results with the kinds of ore usually treated.

The batch for the manufacture of zinc retorts should be prepared with a view to the character of the ore that is to be smelted inside of them. This is a matter upon which a great deal hangs in zinc smelting. If the gangue of the ore (reckoning as gangue everything that is not zinc or in combination with zinc) be basic and the zinc is present as oxide, the clay mixture should also be basic. If, on the other hand, the gangue be silicious or if the zinc be present as silicate, the mixture should be silicious also and quartz may be used in it advantageously for the good qualities which it imparts. It is obvious that if an ore is basic and the retort contains free silica, there will be a tendency to combination of silica with those basic elements lying in close juxtaposition with it at the high temperature at which zinc oxide is reduced. This will occur especially when the ore itself contains elements which unite to form a basic and fusible slag. This drains down the sides of the retorts, collecting in the bottom, and taking up silica from the retort forms a more acid slag. The slag formed in the zinc retort is, however, comparatively small in quantity, and is seldom very fluid.

A selection of clay for zinc retorts is governed by somewhat the same principles as obtain in the manufacture of glass pots. The latter are obliged to withstand the high temperature necessary for fusing the most refractory glass and must have the power of resisting the solvent action of the fused charge, which in glass manufacture is always silicious. According to H. Seger and E. Cramer, this property is possessed in the highest degree by the clays approximating in composition to the formula $Al_2O_3,2SiO_2,2H_2O$.

European Clays.—In order to show the kinds of clay used in Europe for the manufacture of zinc retorts, I have compiled the following table from analyses given in a paper by Doctor Steger, entitled "Die Retorten der Oberschlesischen und der Rheinisch-Westfälisch Zinkhütten":[1]

[1] Zts. f. d. Berg- Hütten- u. Salinenwesen im Preuss. Staate, XLIV, i, pp. 1 to 12.

	I %	II %	III %	IV %	V %	VI %	VII %	VIII %	IX %	X %	XI %	XII %	XIII %	XIV %	XV %
Al_2O_3	39·25	39·93	39·00	39·60	39·76	39·31	32·66	36·75	27·00	29·65	29·00	22·72	28·38	26·70	29·33
SiO_2	44·76	44·88	45·46	44·64	44·87	45·61	50·41	49·00	68·00	53·02	53·86	65·39	58·10	59·18	57·55
MgO	0·36	0·08	Trace	Trace	Trace	.....	.....	0·56		0·78	.....	0·23	.....	0·89	0·26
CaO	0·26	0·21	.....	.....	0·76	0·37	0·50	.....		1·15	0·46	....	.....	1·55	0·74
Fe_2O_3	0·48	0·99	0·011	0·014	1·14	1·13	3·23	0·80	5·00	a3·40	a1·81	0·91	0·57	1·93	2·60
K_2O	1·55	0·52	.....	Tr.	0·67	0·66	1·56	0·41		0·55	.....	0·86	2·49	1·32	.....
Na_2O	.....	.....	.....		.....	.....	.....	0·37		.....	.....	1·84			.....
Loss	13·41	13·03	14·26	14·96	12·95	13·25	11·64	11·87	.....	10·91	14·87	7·77	10 54	8·00	9·33
Totals	100·07	99·64	98·731	99·214	100·15	100·33	100·00	99·76	100·00	99·46	100·00	99·72	99·08	99·57	99·81

	XVI %	XVII %	XVIII %	XIX %	XX %	XXI %	XXII %	XXIII %	XXIV %	XXV %	XXVI %	XXVII %	XXVIII %
Al_2O_3	27·23	43·00	36·69	42·69	44·92	38·30	45·21	46·39	27·62	34·78	29·53	19·49	17·62
SiO_2	61·69	54·70	49·42	55·73	54·00	b43·84	52·50	51·87	51·25	c49·64	57·95	71·79	73·58
MgO	.....	.....	Trace	0·23	Trace	0·19	0·54	0·45	0·45	0·41	0·36	0·06	0·40
CaO	0·38	0·30	0·09	0·28	0·48	0·19	.....	.....	0·23	0·68	1·72	.....	1·24
Fe_2O_3	1·24	1·70	0·73	0·37	0·80	0·46	0·81	0·52	d1·14	1·80	1·88	0·85	1·29
K_2O	.....	0·30	0·43	0·50	0·60	0·42	0·51	0·22	1·89	0·41	.....	0·85	.....
Na_2O	.....	.....	.....	.....	.....	.....	.....	.....	0·85	.....	.....		.....
Loss	9·61	.....	12·70	0·43	.....	17·78	0·78	.....	14·75	12·00	8·45	7·47	5·50
Totals	100·15	100·00	100·06	100·23	100·80	99·18	100·35	99·55	100·22	99·72	99·89	100·91	99·63

(a) FeO. (b) Quartz, 4·90%; chemically combined, 38·94%. (c) Quartz, 9·95%; chemically combined, 39·69%. (d) This sample in addition contained 2·04% FeO.

I, Briesen clay, average sample, Bischof; II, Briesen clay, purest sort, selected with magnifying glass, Bischof; III, Briesen clay, analyzed in laboratory of Paulshütte, Schoppinitz, O. S.; IV, ibid; V, Briesen clay from Ferdinand shaft, analyzed at the experimental station of the Royal Porcelain Factory at Berlin, 1888; VI, Briesen clay from Anton shaft; VII, Saarau clay, Kerl; VIII, Saarau clay, analyzed at Paulshütte, O. S.; IX, burned clay from Striegau; X, gray clay from Striegau; XI, dark blue clay from Striegau, near Saarau; XII, clay from Grojec; XIII, kaolin from Saarau, analyzed at Paulshütte, O. S.; XIV, ibid, by R. Schumann; XV, kaolin from Pentsch; XVI, kaolin from Ruppersdorf, near Strehlen; XVII, kaolin from Ruppersdorf, puddled and burned; XVIII, kaolin from Göppersdorf, dried at 100° C.; XIX, burned shale from Neurode, analyzed by Aron; XX, ibid, by R. Schumann; XXI, raw Neurode shale, Bischof; XXII, shale from Rakonitz, Bohemia, analyzed by Aron; XXIII, ibid, by Seger; XXIV, shale from Morgenroth mine at Schoppinitz; XXV, blue clay from Andenne, Bischof; XXVI, black clay from Andenne, Schmidt; XXVII, clay from Namur; XXVIII, clay from Andenne, Bischof.

Analyses I and II are due to Bischof, Die feuerfesten Thone, p. 181; V and VI, Hecht, Thonind. Ztg., May 26, 1888; VII and XXVII, Kerl, Thonwaaren Industrie.

I-VI. Briesen clay possesses a quotient of refractoriness of 60%, fuses with Seger cone No. 35, and is therefore exceeded by no other clay in infusibility; only the Neurode, Rakonitz and other shales (Thonschiefer) and some puddled kaolins stand higher. A rational analysis of V and VI showed in the former, pure clay (Thonsubstanz) 99·07%, quartz 0·32%, and feldspar 0·61%; and in the latter, clay 99·67%, and feldspar 0·33%. This clay comes in stone-hard lumps; it is ground to 1 mm. size and is then wetted and allowed to rot several months, after which it forms excellent retort material.

VII, VIII. Saarau clay, which comes in large wet clods (with 20 to 30% water), is a fat, strong-binding clay, but stands considerably behind the Briesen in infusibility and non-corrosibility with slags. About 14% of its silica is present as sand, the remainder being chemically combined.

IX-XI. The Striegau clay is plastic, but inferior in infusibility to the Saarau. It shrinks much in burning, as do both the Saarau and Briesen.

XII. Grojec clay is a non-shrinking material, which is mixed with the preceding varieties to counteract their contraction. This clay is always used raw. At moderate temperature it burns dense and firm; at white heat it fuses. It can only be used, therefore, in very small proportion.

XIII-XVIII. These kaolins have been used as a substitute for the Grojec clay. They are quartzose and very refractory; to reduce their sand contents they are sometimes slimed. They are very lean and fragile. In the fire they shrink until their chemically combined water is driven out, but then expand on account of their high quartz content, and withstand very high temperatures without fusing, though on this account they are susceptible to changes of temperature. Their high silica tenor, however, as is the case with the Grojec clay also, makes them likely to slag with the basic material of the charge. On account of their leanness these kaolins contribute practically nothing to the binding of the mass. To use them raw also contravenes their utility, since they would serve their purpose much better if burned to chamotte, when more fat raw clay could be used, while the kaolin-chamotte would be a component promoting the infusibility of the mixture. The coarse grains of free quartz will still prove a defect in these kaolins, however, since they break under rapid variations of temperature, to which the retorts are subject in the furnace. The slimed kaolins, which seem best adapted to replace the binding clay, have in practice proved to be entirely too lean.

XIX-XXIV. The Neurode shales and equivalents have proved the best lean material for admixture in the retort mass in Upper Silesia, being cheaper than chamotte, and superior in infusibility and non-corrosibility with slags. Together with the shale, old retort material dressed clean from adhering slag is used. The use of the latter limits the temperature in the Silesian furnace below that which is commonly attained in the Rhenish-Westphalian, where but little of it is employed. The Neurode burned shale, which is used in Silesia as a partial substitute for the old material, to increase the refractoriness of the batch, and counteract softening, is an excellent material, fusing with the highest Seger cone, No. 36. The particles of shale hold out amazingly in the mass; only their surfaces are acted upon chemically, while their interiors remain unaltered. Even after being used

several times, first by direct admixture and subsequently in the old material, they are still highly refractory.

XXII-XXIII. The Rakonitz shale is equal in quality to the Neurode, but is more expensive.

XXIV. A shale from the coal measures at Schoppinitz, which proved ill adapted as a substitute for the Neurode on account of its high fluxing components.

XXV-XXVIII. The clay from Andenne, near Namur, Belgium, occurs as two distinct kinds: The blue to blue-black, less silicious, extraordinarily fat and binding; the light-colored, very silicious, less fat. The former is used raw and the latter (for lean material) burned. The Andenne fat clay corresponds to Seger cone No. 33.

Other analyses of European clays are as follows:

	XXIX %	XXX %	XXXI %	XXXII %	XXXIII %	XXXIV %
Al_2O_3.....	26·00	24·00	39·76	39·68	20·00	18·00
SiO_2.....	54·00	63·00	44·87	45·11	71·00	76·00
MgO.....	....	....	tr.	tr.	....	....
CaO.....	....	1·00	0·76	0·07	....	....
Fe_2O_3.....	2·00	1·00	1·14	1·25	2·00	tr.
K_2O.....	....	....	0·67	0·66	....	....
Na_2O.....	....	....	....	....	....	....
H_2O.....	20·00	11·00	12·95	13·24	7·00	6·00
Total.	102·00	100·00	100·15	100·01	100·00	100·00

XXIX, Clay from Andenne, Belgium, Kerl Grundriss der Metallhüttenkunde, 451; XXX, Clay from Namur, Kerl, idem; XXXI, Clay from Briesen, near Lettowitz, Moravia, analysis by Hecht, Kosmann, Oberschlesien, etc., 206; XXXII, Clay from Lettowitz, Moravia, analysis by Hecht, Kosmann, op. cit.; XXXIII-XXXIV, Clay from Natoye, Belgium, Schnabel, Handbuch der Metallhüttenkunde, II, 99.

American Clays.—The zinc smelters of the United States use clay from either Woodbridge, near Perth Amboy, N. J., or Cheltenham, a district of St. Louis, Mo.,[1] the latter being freighted so far as Pulaski, Va. Analyses of these clays are given in the following table:

[1] Since this chapter was put in type the results of an extremely elaborate and highly scientific investigation of the properties of St. Louis clay, especially with regard to its employment for the manufacture of zinc retorts, have been published by Doctor Otto Mühlhaeuser in Zts. f. angew. Chem. XVI, 148 to 159, 222 to 225 and 273 to 282. Doctor Mühlhaeuser reports the following average analysis of this clay, dried at 120° C.: Al_2O_3, 34·46%; SiO_2, 49·50%; Fe_2O_3, 2·39%; CaO, 0·80%; MgO, 0·62%; loss on ignition, 12·86%. Its composition is quite uniform. Its specific gravity is 2·56, and it fuses between Seger cones 30 and 31. In point of refractoriness it is therefore an entirely suitable material for retort manufacture, but unfortunately it does not burn as dense as is desirable; after annealing the retorts have a porosity of 28 volume-per cent., and after burning 24·5 volume-per cent. This leads to important losses of metal by absorption in the walls of the retorts and filtration through them. Doctor Mühlhaeuser has made an exhaustive study of the behavior of the retorts under all conditions,

	I %	II %	III %	IV %	V %	VI %	VII %	VIII %	IX %	X %	XI %
Al_2O_3..	31·53	30·08	30·47	25·64	31·68	35·90	37·32	37·01	39·53	35·83	39·24
SiO_2...	*a* 50·80	50·19	50·16	61·02	56·01	*e* 46·90	42·85	42·83	*e* 42·73	*g* 47·75	*i* 43·41
MgO...	....	0·47	0·29	0·08	0·21	....	0·41	0·46	....	0·11	....
CaO...	....	1·31	1·51	0·70	1·17	....	1·48	1·41	0·10	....	0·20
Fe_2O_3..	1·92	2·79	2·48	1·70	1·13	1·10	1·18	1·04	0·50	0·77	0·46
K_2O...	0·40	0·65	0·97	} 0·73	0·09	0·28	0·76	0·85	} 0·49	} 0·44	} 0·89
Na_2O..	....	....	....			0·16	....	....			
TiO_2...	1·50	....	....	....	....	1·30	....	....	1·40	1·10	1·60
H_2O...	*b* 13·80	14·65	14·62	10·00	9·71	*d* 14·30	16·36	16·27	*f* 14·80	*h* 13·70	*k* 14·90
Total.	99·95	100·14	100·50	99·87	100·00	99.94	100·36	99·87	99·55	99·70	100·70

a, Free silica (quartz), 12·70%; *b*, combined water, 11·30%; *c*, free silica, 6·40%; *d*, combined water, 12·80%; *e*, free silica, 0·50%; *f*, combined water, 13·59%; *g*, free silica, 5·70%; *h*, combined water, 12·20%; *i*, free silica, 0·70%; *k*, combined water, 13·32%.

I, Clay from Cheltenham, Mo., E. C. Moxham, Eng. and Min. Journ., Nov. 25, 1893, p. 544.
II, III, Clay from Cheltenham, Mo.
IV, Clay from Cheltenham, Mo., H. Ries, The Mineral Industry, II, 209.
VI, Clay from Woodbridge, N. J., Moxham, loc. cit.
VII, VIII, Clay from Woodbridge, N. J.
IX, X, Clay from Woodbridge, N. J., Ries, loc. cit.
XI, Clay from South Amboy, N. J., Ries, loc. cit.

Chamotte.—Chamotte, or burned clay, commonly referred to by American zinc smelters under the ambiguous term "cement," is prepared by grinding raw clay to pass a screen with 1 to 3 mm. holes, kneading with addition of water in any suitable pug-mill, cutting up the dough ejected from the mill in more or less rectangular lumps ("adobes"), drying and burning in an ordinary kiln.

Old Material.—The material of old retorts dressed clean from adhering slag is generally used in preparing the batch for new retorts, serving the purpose of chamotte than which it is much cheaper. It is indeed a chamotte which has undergone a complete change in the furnace through the action of the zinc vapor absorbed in it, which with the alumina forms zinc spinel and tridymite. This is manifested by a blue coloration of the clay of old retorts, which is sometimes a deep purple. Degenhardt reported the following analyses of old retorts from Bethlehem, Penn.[1]:

Color.	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	ZnO %	MnO_2 %	CaO %	MgO %	Totals. %
Blue....	41·13	33·48	2·84	21·47	0·37	0·92	0·47	100·68
White..	50·10	38·28	3·42	6·10	0·41	1·13	0·73	100·17

Old material is of limited refractoriness; retorts containing a high percentage of it tend to bend and break, and those in which it is replaced by

[1] American Chemist, 1875, p. 355.

good chamotte are significantly stronger. Retorts which are designed to withstand a very high temperature should be made without admixture of old material in the batch, or at least but comparatively little of it should be employed.

The percentage of zinc in old retorts is usually from 6 to 10. Analyses at the Paulshütte, at Schoppinitz, Upper Silesia, showed the following: Light green, 6·45%; light blue, 8·55%; dark blue, 9%. The loss of zinc which is incurred in this manner is discussed in Chapter XII.

PREPARATION OF THE BATCH.—In preparing the batch the clay and chamotte are first ground by suitable machines, roller mills being most commonly employed in Europe and slow-running, geared rolls in the United States. There are works in Europe which pulverize the clay with grinders of the coffee mill type and with the Vapart centrifugal mill, while certain works in the United States use roller mills. Probably the roller mill, or "dry pan," as it is commonly called in the American clay-working industry, is the most efficient machine for grinding clay. There is no especial difficulty in this process, however, since the clay comes usually in dry, stony form, which is easily pulverized, and of course there is no difficulty in grinding the chamotte and old material.

The rolls employed for clay-crushing require no description, inasmuch as they are precisely analogous to the machines employed for crushing other soft minerals. A form of roller mill made by C. Mehler, of Aachen, Germany, which is used by European zinc smelters, is illustrated in Fig. 170. This machine is made in three sizes, of which the data are as follows:

No. of Mill	Rollers				Capacity kg. per hour	*b* Driving Pulley		H. P. required	*c* Floor Space	Weight Complete kg.
	Diam. mm.	Width mm.	*a* Weight kg.	r.p.m.		Width mm.	r.p.m.			
1	1,500	400	3,400	10	1,500	210	42	8	3·25 × 2·50	13,500
2	1,250	320	1,750	12	1,000	160	50	6	2·25 × 1·75	9,000
3	1,000	260	1,000	15	500	125	64	3	2·00 × 1·60	4,600

a Weight per roller; *b* The diameter of the driving pulley is the same as that of the roller in machine; *c* Dimensions in meters, the first being the length and the latter the width in each case.

The Vapart centrifugal mill, which is also used extensively in Europe, is described and illustrated in Chapter XI. In clay-crushing in Belgium these machines deliver about 3000 kg. of product per hour.

Degree of Crushing Required.—The clay is ground to a meal, usually not to exceed 2 mm. size, but the chamotte and old material are not made so fine, being crushed only to pass a 5 mm. hole or about that size. If all the

material were fine, the retort would be weak and would bend easily in the furnace. On the other hand, if the grains of sand, chamotte, etc., in the mixture are too large, the retort is apt to be porous and zinc vapor will be lost by penetration of its walls. Too large a proportion of chamotte also makes the retort porous and moreover fragile and hard to form. At many

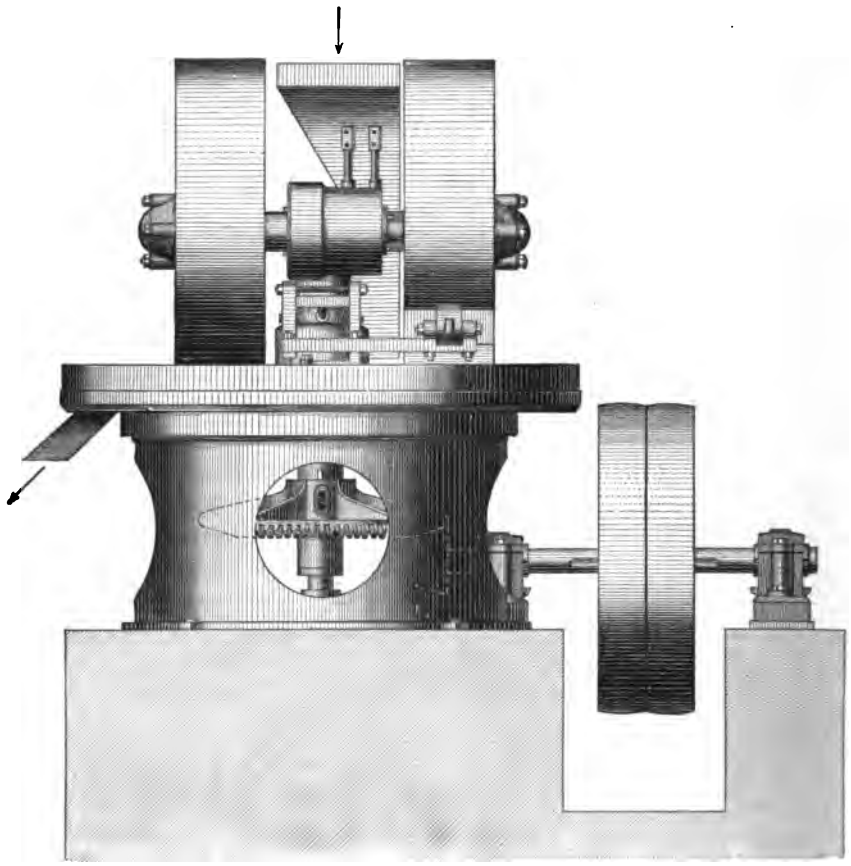


FIG. 170. ROLLER MILL, FOR CLAY CRUSHING.

American works the chamotte is crushed much finer than indicated above, e.g., at Iola both raw clay and chamotte are ground to 1·5 mm. size. In Upper Silesia on the other hand chamotte is used in grains as coarse as 9 mm. and even 10 mm.

It is of great importance to select a proper size of grain for the chamotte

which is to be used in the fabrication of the retorts. Its function in the batch is to reduce the plasticity of the raw clay, to prevent too much shrinkage and to make the evaporation of moisture uniform. The chamotte forms a kind of skeleton around which the particles of raw clay are uniformly distributed if the mixture is complete (and that is essential) acting as a center of shrinkage for the raw material surrounding it. The chamotte also possesses the power to absorb water which during drying is returned again to the raw clay, and in that manner the formation of cracks and fissures is prevented. The larger the size of the chamotte particles the better are the retorts able to withstand changes of temperature, and the greater their resistance to chemical corrosion owing to the less surface presented to attack. Between these advantages and disadvantages a middle course is to be selected which will give the best average result.

Proportions of the Constituents.—In general the proportion of chamotte and raw clay employed varies from equal parts of each to 60% chamotte and 40% raw clay. It is a common practice to make up the chamotte one half of newly burnt clay and one half of old material, but some smelters do not use any old material at all. The American and European practices do not vary much as to the proportions of raw clay and chamotte where those are the only constituents employed in the batch. It is rare to find the percentage of raw clay less than 40, although as little as 35% is reported to have been used under certain conditions. At the present time the clay of the batch for retort manufacture in Belgium, Rhenish Prussia and Westphalia is apt to be made up with more ingredients than the essential constituents (raw clay and chamotte), the additions being chiefly coke dust and sand; both those substances being lean-making material, they are substituted for a part of the chamotte. An addition of coke dust, up to 10% of the batch, is now commonly made. It imparts to the walls of the retort a peculiar smoothness and greater density, giving them the appearance of a graphite crucible, and apparently tends to prevent the absorption of zinc vapor and diminishes the corrosive action of slags and metallic oxides. Opinions as to the action of the coke dust vary, but whatever it is there is no doubt that coke is an excellent lean-making material, not shrinking even at white heat, and to it is also due that so much fine chamotte may be used without danger. As to the use of coke Doctor Bischof says:[1] "Anthracite, coke, and even charcoal dust, are more easily ignited than graphite, yet so long as they are protected from burning by a clay envelope they increase the refractoriness of the clay and counteract the tendency to crack."

There has been lately a tendency in Belgium to use a considerable per-

[1] Die feuerfesten Thone, p. 206.

centage of sand in the batch, especially when the retorts are to be employed for the distillation of a silicious ore. Retorts made of a mixture rich in silica cost less than those made from mixtures richer in clay and can be molded with thinner walls (as thin as 20 mm.)[1]

According to Doctor Foehr[2] fluorspar has been used experimentally in the batch for the preparation of retorts, a mixture of quartz sand with about 3% of fluorspar and some soda being recommended. Retorts made of that mixture frit on the surface during annealing, but after the escape of the fluorine become far more dense and fireproof than otherwise can be produced. In fact, a small quantity of fluorspar is used regularly in potteries which make fireproof ware. Zinc retorts are said to be improved very much by baking with a glaze consisting of equal parts of fluorspar and zinc sulphate.

The mixture of clays, etc., varies considerably among the Belgian-Rhenish-Westphalian works. One uses 40 parts by measure of dark, fat, raw Belgian clay; 50 of light-colored, sandy Belgian clay burned to chamotte; and 10 of coke. Another 36, 54 and 10 parts of the same substances respectively. Both are very durable. A third uses 30 parts of fat, Belgian clay (raw); 10 of sandy Belgian clay (raw); 50 of old retorts, and 10 of coke, which has less durability on account of the old material. A Belgian works used 10·5 parts Tahier clay (raw); 10 of the same burned; eight of old material; one of quartz, and 0·5 of coke. In the last case the quantity of coke is too little to be of influence. At Dortmund, Westphalia, the batch is made up of one part of fat clay from Andenne and two parts of burned gray clay from Andenne; to that mixture 10% by volume of coke dust is added.

The average mixture of clays used in Upper Silesia, varying somewhat at each works, is: Fat clay (raw), two parts by measure; sandy clay (raw), one part; lean material (old retorts and shale), five parts. As fat clays are used the Briesen, Saarau and Striegau, each alone or a mixture of two or three; as sandy clay the Grojec or Silesian kaolin, or a mixture of both; as lean material old retorts and shale in varying proportions. The clays are sifted through 1 to 3 mm. screens; the old material and shales are used as a mixture of coarse (up to 9 mm.) and fine, just as is delivered by the roller mill, which is commonly employed for grinding them.

METHODS OF MIXING.—The constituents of the batch having been pulverized separately to the required size, they are mixed in the predetermined proportion, either by shoveling on a floor or by a mechanical mixing machine and are then kneaded with the proper proportion of water necessary

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 114.
[2] Chem. Ztg., 1890; Mineral Resources of the United States, 1889-90.

to develop the plasticity of the clay, which is usually about 8 to 10%, but may be as high as 20%, in some suitable machine, usually a vertical or horizontal pug mill. The old method of mixing the clay by treading is no longer practiced. After the first pugging the batch is deposited in a heap, in which it is allowed to remain for two to six weeks, during which time it is kept covered with wet cloths. This tempering or rotting is to develop further the plasticity of the clay. At the end of the tempering period the clay is drawn from the pile as required, pugged a second time and then delivered to the retort molders. This process is followed generally in Europe, but there are few zinc smelters in the United States who go through so much detail, the common practice in Missouri and Kansas being to mold the retorts immediately after the preparation of the batch. Leaving

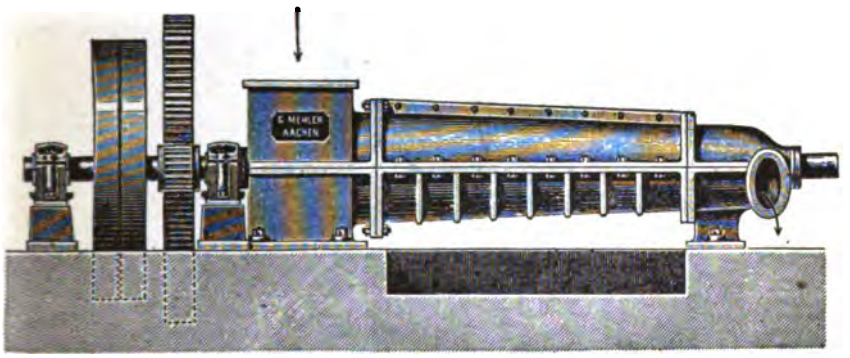


FIG. 171. HORIZONTAL PUG-MILL.

the clay to rot after the first pugging implies of course a rehandling of the entire batch and adds correspondingly to the cost of the retorts.

Pug-Mills.—Types of pug-mills employed for kneading clay are shown in the accompanying engravings. In the potteries of zinc smelting works the horizontal machines with closed tops are most commonly used. The machine shown in Fig. 171 is employed in Germany. A horizontal pug-mill used in the United States has a cylinder made of ¼ in. steel plate, the top being closed with removable, sectional cast-iron covers. At the discharge end there should be an adjustable gate to regulate the discharge of the pugged clay. The shaft is 3¾ in. hexagonal, hammered steel. The mixing knives are of steel, fitted into socket hubs which are slipped on the shaft and fastened by bolts. The knives are capable of being set in different positions according to the work that is to be done. If the clay requires thorough pugging the knives are set at a small angle, so that the

clay is slowly forced forward; if less pugging is required and greater capacity is needed, the angle of the knives is increased. Obviously the degree of pugging depends also upon the length of the cylinder. A common length is 8 ft., but smaller machines are frequently employed where the quantity of clay to be pugged is small.

MOLDING THE RETORTS.—Retorts were formerly made universally by hand, the walls being built up and solidified with proper tools in wooden molds of the requisite shape. The large retorts used in Upper Silesia are still made by hand, the peculiar shape which has been adopted there making

FIG. 172. VERTICAL PUG-MILL.

it difficult to form them mechanically.[1] The manufacture of the Belgian and Rhenish retorts, on the other hand, is now done all but universally by machine, both in Europe and the United States.

Augur Machines.—In Kansas, Missouri and Illinois zinc retorts are commonly made with the aid of a machine similar to that which is employed for the manufacture of drain pipe. The ordinary machine of this type costs $750 at the works of the maker, freight and erection bringing the cost to the zinc smelter to approximately $1100. With such a machine a retort maker and two helpers are able to make 80 retorts per day.

[1] One works in Upper Silesia has recently tried pressed muffles, but these are small ones, such as are used in the Rhenish furnaces.

The arrangement of such a machine at Lasalle, Ill., is illustrated in the accompanying sketch. The vertical cast iron cylinder is supported by the beams of the upper floor and the bracing rods as shown. The clay, raised from the lower floor by the belt elevator, is dumped into the top of the cylinder, wherein there is revolved a vertical shaft bearing blades set at a suitable angle, which press the clay downward and force it out at the bottom, around a core, which corresponds to the internal diameter of the retort, and through a die-plate agreeing with the outside diameter. At the top of the cylinder the shaft is provided with a scraper to dislodge any clay that may stick in the hopper. The pipe of clay issuing downward through the die is received on a counterbalanced pallet, which sinks as the

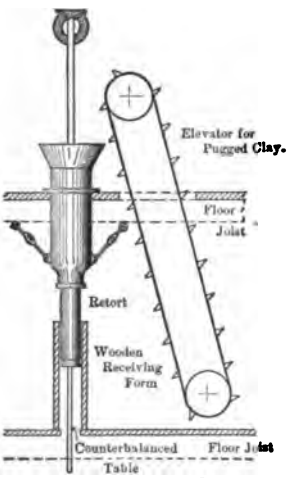
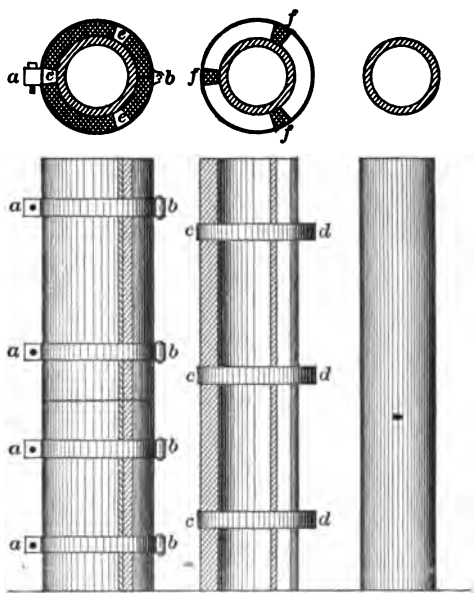


FIG. 173. ARRANGEMENT OF AUGUR MACHINE FOR MAKING RETORTS.

pipe descends. This is designed to steady the pipe as it emerges from the die and cause it to run smoothly. The retort is received, moreover, inside of a portable form, within which the pallet rises and falls. When a clay cylinder of suitable length has been made, the machinery is stopped and the cylinder is cut off at the top of the receiving form by means of a wire hanging from a convenient place. The receiving form is then removed on a truck, and a cylinder of clay about 8 in. in diameter and 2·5 in. thick is tamped in solidly to form the butt of the retort.

The receiving forms are made of wood in two sections, a long one and a short one, and each is divided into three segments, which are held together by the engirdling rings, but are separated by spaces, as shown in the accom-

panying sketches. In the openings between the segments loose staves equal in length with the retort are inserted. In removing the form from the finished retort the upper section is first opened, the three staves remaining in position since they are still held by the lower section. Two iron rings are then slipped over the staves and the lower section is removed, after which a third ring is put on and the others are shoved down, as shown in Fig. 178. The object of this device is to preserve the shape of the retort during the critical period of the first two or three days' drying. The retorts dry at the normal temperature of the air for 10 or 12 days, care



FIGS. 174 TO 179. FORM FOR HOLDING BELGIAN RETORTS.
Plans and elevations.
Legend: *a*, clamps; *b*, hinges; *c d*, iron rings; *e*, spaces between sections of form; *f*, wooden staves.

being taken that no draught strikes upon them which would make them crack, or at any rate would be apt to weaken them. They are then placed in the hot room, where they are kept at least 50 days, the green retorts containing about 16% of free moisture, to be dried out as thoroughly as possible.

This system of making retorts is practiced generally in the United States, with unimportant modifications, save at the few works which employ hydraulic presses. The vertical augur machines are the most common, but the combination of a horizontal pugging cylinder with a vertical pressing

cylinder, as in many modern drain pipe machines, is also used. The method of setting the pressure blades or augur varies in the machines of different manufacturers. So too does the method of supporting the core. In some machines the latter is held in position by straight bridge-trees connected with the main cylinder. With such an arrangement the clay is necessarily split in running over the bridge trees and although it is united again in the die below them there are likely to be lines of weakness caused by the separation of the clay and imperfect reunion, which is manifested by cracks developing in the retorts during drying. This difficulty is lessened by making the bridge-trees curved in plan, instead of straight. Other machines dispense with bridge-trees, the core being supported by the augur shaft so that the annular mouth of the machine is free from obstacles.

Boring Out Machines.—At the works of the Société Anonyme de la Vieille Montagne at Angleur, Belgium, the thoroughly kneaded clay is hammered solidly into a canvas sack which forms the lining of a cylindrical or elliptical mold of the shape and size of the desired retort, and the column of solid clay which is formed in that manner is subsequently reamed out by a vertical boring machine. The molds which are made of iron open longitudinally to discharge the finished retort, the two halves when in use being clamped together with screws through the lugs of the bands which surround them. The clay is packed solidly into the molds by means of a hammer working up and down by a simple mechanical contrivance. When filled solidly the mold and its contents are transferred on a truck to the boring machine, which cuts out an elliptical or circular hole as required. The boring of a circular hole is so simple that no special explanation is necessary. In making elliptical retorts the mold has of course the requisite form for their exterior, while the interior is cut out by parallel borings intersecting each other throughout their whole length. The retort having been bored out, it is removed from the mold and sent to the drying house, wherein the canvas sack enclosing it is stripped off and the inside is made smooth by hand, after which it is set aside to dry for a few days at the temperature of the atmosphere, and finally is placed in the hot room.

The machines above described were introduced at Angleur at least 40 years ago, having been described by Percy in his *Metallurgy,* 1861, p. 583, who stated that retorts made with them cost at that time 1·6 fr. (32c.) apiece, while at Moresnet in 1857 the cost of hand-made retorts finished and burned was 2@2·1 fr. (40@42c.) apiece, one man being capable of making 18 to 20 retorts by hand in 12 hours. The most important saving effected by the machine-made retorts was, however, because of their greater durability and the increased recovery of zinc made possible with them. In

1840 the consumption of retorts at Angleur was 11·9 to 25 per 1000 kg. of zinc (average 17), and in 1888 it was only one tenth as much.[1]

Hydraulic Press Machines.—A good many of the Belgian and Rhenish zinc works have now adopted hydraulic presses for making retorts, and there are several works in the United States in which machines of that type are used. The clay is prepared for the hydraulic machines, after a thorough kneading, by compression into cylinders (ballots) about 0·3 to 0·4 m. in diameter and 0·4 to 0·5 m. long, varying according to the size of the retort that is to be made. The ballots are prepared by mechanically hammering into a mold, of which the bottom is an hydraulic piston, lumps of the soft homogeneous clay, which are cut from the sausage (about 0·25 m. in diameter) ejected by the pug-mill. The clay is fed into the hammering machine little by little and the block is kept up to meet the blows of the drop hammer by the piston, which is lowered gradually as more clay is fed in. When thoroughly solidified, the cylinder is shoved out of the mold by raising the piston so that it can be pushed on the table at one side. It is then ready for the hydraulic press. The hydraulic presses for making zinc retorts, which were first introduced by E. Dor, a Belgian engineer, and were used at the Asturienne works at Auby, France, as early as 1877, are of two kinds.

Bottom Feed Type.—In one kind of hydraulic press a proper block (ballot) of clay is forced by an hydraulic piston into a vertically standing steel cylinder, the inside of which is of the same shape and size as the outside of the retort to be made. Within the cylinder there is a steel core of the same size as the inside of the retort. The clay having been forced from below into this mold, around the core, under a pressure of 150 to 200 atmospheres, the cap of the cylinder is removed and the formed retort is pushed upward and out by raising the hydraulic piston. The retort is then cut by a wire at the proper length, received on a carriage and sent to the drying house. A chunk of clay is thrown from above into the cylinder to make the bottom of the next retort, the cap of the cylinder is replaced and locked on securely, a block of clay is put in at the bottom and the pressure is exerted as before. The method of feeding in the clay in this type of press is rather clumsy. The main cylinder stands about 0·5 or 0·6 m. above the feed table and is open at the bottom. The core inside of it is supported by bridge-trees. The ballot of clay is put in a cylinder, open top and bottom, of the same diameter as the upper cylinder, with which it makes a close connection when put in place beneath it. The piston pushes the clay up through the lower cylinder. The latter is then removed and

[1] Bulletin de la Société de l'Industrie Minerale, 1888, p. 505.

another one which has previously been made ready is put in its place. The ballots of clay are prepared on the lower floor of the pottery, where the pug-mills are, and the molded retorts are delivered on the upper floor.

Top Feed Type.—The hydraulic press which is now generally in use in Belgium, Rhenish Prussia and Westphalia is fed from the top; although in some respects it is less simple than that which has been described above, it is easier to operate and is more efficient. It consists of a heavy steel cylinder, supported vertically by columns standing on a suitable pedestal. Operating inside of the cylinder there are two rams, one of them being annular and fitting to the main cylinder, and the other being cylindrical, working inside of the annular ram. Pressure is exerted on both of the

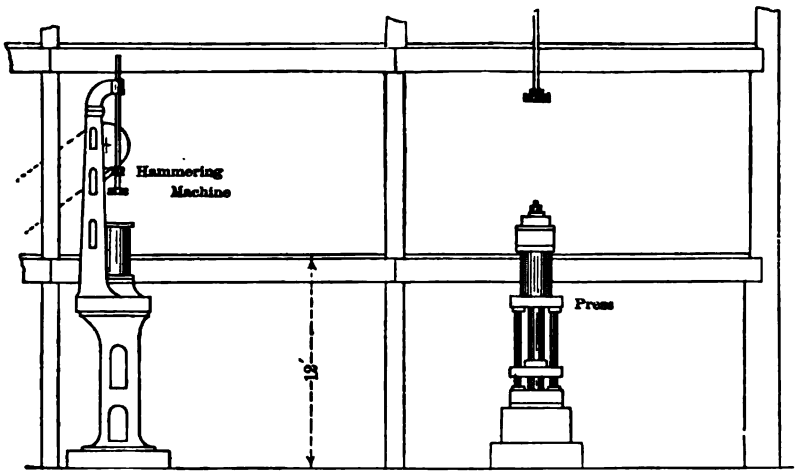


FIG. 180. ARRANGEMENT OF HYDRAULIC MACHINES FOR MAKING RETORTS.

rams from below. The cylindrical ram being central to the machine, its own base serves as the piston. The annular ram is connected with a heavy yoke which is raised and lowered by two pistons, working together, one on each side. The cylindrical ram passes through this yoke. Of the four columns which support the main cylinder, two diametrically opposite extend above the top of the latter; the other two extend only a little above the bottom, passing through lugs, which are bored out for them. A heavy nut on the end of each column holds the cylinder down securely. The top of the cylinder is closed first by a heavy die block, the shape of the exterior of the retort, and above that a cover plate, both of which are hinged on one of the columns and can be swung horizontally aside by means of levers; when in proper position they lock under the nut of the opposite column.

The general proportions of such a machine, together with the accompanying hammering machine, are shown in the accompanying sketch.

In making a retort, a ballot of clay is put into the cylinder from the top, both the die block and the cover plate being removed. The latter are then closed and locked and both rams together are moved upward, pressing the clay solidly into the cylinder. The inner ram is then forced into the clay, while the annular ram, still under pressure, retreats as the clay is crowded backward around the inner ram. When the clay has been thus molded, the cover plate is pushed aside and the retort is thrust up and out through the die by raising the annular ram. The portion of the cylindrical ram which projects inside the die has of course the shape of the interior of the retort, the butt of which is pressed between the ram and the cover plate.

The pressure under which the retorts are formed varies from 150 to 200 atmospheres, 180 to 200 being the commonest practice. This is developed by a suitable pump. The clay is under that pressure for about two minutes. The capacity of a machine is 75 to 85 retorts per 10 hours. Its operation requires two men, while a third man attends to the preparation of the ballots of clay with the hammering machine and a fourth removes the finished retorts to the drying room. This does not include the labor for crushing the clay and preparing the batch, which is likely to require two men in addition.

Cost of Hydraulic Press Plant.—A plant for the manufacture by hydraulic pressure of 75 to 85 retorts per 10 hours comprises the following machines:

1. Pug mill, requiring 20 h. p.
2. Elevator for delivering the pugged clay to the clay-hammering machine.
3. Clay-hammering machine.
4. Hydraulic retort press.
5. Hydraulic pump.

The hydraulic machines require about 5 h. p. in the aggregate. The cost of those machines at the manufacturers' works at Aachen, Germany, in 1894 and 1900 was as follows:

Machine	Weight Kg.	Marks 1894	Marks 1900
1. Pug mill	5,700	3,000	3,360
2. Clay hammering machine	5,000	2,600	2,950
3. Retort press	8,000	4,600	5,340
4. Hydraulic pumps	3,700	3,150	3,400
Total	22,400	13,350	15,050

It will be observed that an hydraulic retort plant is considerably more expensive than a plant using the ordinary augur machines which are em-

ployed generally by American zinc smelters. Nor are the retorts made any more cheaply than with the augur machines; indeed, they may be more costly. The advantage to be derived from the use of retorts made under hydraulic pressure is due entirely to their superior quality, their greater density reducing the loss of zinc by absorption and penetration, while their greater density combined with their greater strength renders them more able to resist the corrosion of slags and makes them capable of withstanding a higher temperature and also of being made with thinner walls, with the result that the retorts have not only a longer life, but the charge can be "pinched" more severely to extract the last of the zinc and the loss of zinc in the residues is consequently reduced.

Retort Molding by Hand.—In Upper Silesia the muffles are made invariably by hand, the method employed being the same as has been frequently described in the textbooks. The workman begins by plastering a sheet of clay on the bottom and sides of a wooden form, which is one third of the length of the muffle and is made in two sections so as to open longitudinally, the two halves being held together with clamps. The clay which is plastered in is beaten with a wood paddle against the sides of the form in order to give it the requisite density, the seam being carefully united. The first section having been finished, another similar form is set on top of it, and the retort is built up inside of that, after which a third section is put in place and the retort completed. When the retort has been finished, the sections are removed by unscrewing the clamps which hold them. Retort manufacture in this manner is slow and laborious, but the retorts thus made serve their purpose well enough in the Silesian process and there is no apparent tendency to adopt the expensive machines which are now used in Rhenish Prussia and Belgium. The Silesian metallurgist is not inclined to abandon the tapering sides of his muffle, which it would probably be difficult to make with a press, and there is not the same requirement of strength for them as for the Belgian and Rhenish retorts. The cost of a Silesian, hand-made muffle is said to be 4@5 marks (about $1@$1·25). One man makes 20 to 24 per week of six days. He usually has from four to five in hand at the same time, working at intervals upon each of them in their different stages of manufacture, which saves time and economizes labor to some extent.[1] The manufacture of four muffles per day would imply a labor cost of 20@25c. per muffle.

Belgian retorts are made by hand in a manner quite similar to the Silesian. In Europe it used to be reckoned that one man could make 18 to 20 per 10 hours, the cost in Belgium being 1·75@2 fr. (35@40c.)

[1] Max Georgi, Berg- u. Hüttenm. Ztg., March 2, 1877.

apiece, including the clay. At a small works in Kansas in 1899 a man made 15 and a man and a boy 20 per day, including the mixing of the batch. The contract price was 16c. apiece (for labor only). It will be observed that hand-made retorts are not much more expensive than those which are made by the augur machine, while with the docile Joplin ore at least they are said not to be materially inferior in durability. The superiority of the machine-made retorts is chiefly in their greater homogeneity and density, which reduce the loss of zinc by absorption and seepage.

Cost of Making Retorts.—The labor expense for making retorts does not vary greatly in different works; the cost for clay is of course governed by the situation of the works with respect to that material. One retort maker with two helpers, working 300 days per annum, can easily make all the retorts required for a works with six Belgian furnaces, smelting 12,000 tons of blende per annum and producing approximately 6000 tons of spelter. When working with the machine they will make 75 to 80 retorts per day and including the mixing of the batch the monthly average will be 45 to 50 per day. In larger works the machine will be kept in operation for more of the time and additional men will be provided for preparation of the batch. Thus a single retort machine is sufficient for the needs of a large smeltery; nor is it necessary to install the pottery machinery in duplicate, since the stock of retorts that must be carried any way is so large that a break-down of even a week would not cause any inconvenience.

In Kansas it is common to estimate the value of a seasoned retort at 50c. and that of a condenser at 3@4c. The expense for labor is about 15c. per retort; the clay cost varies according to the weight of the retort and the proportions of raw clay, chamotte and old material that are employed. The raw clay costs $3 per 2000 lb. delivered, and about 25c. per retort is representative of the material in a single retort. Labor for rehandling, steam for power and heating, repairs and renewals of machinery and building, miscellaneous supplies, etc., further increase the cost.

Manufacture of Condensers.—The batch of refractory material for the manufacture of condensers is prepared in the same manner as for the manufacture of the retorts, but the clay need not be of so good quality, inasmuch as the condensers are obliged neither to withstand a high temperature nor the action of corrosive slag; they are subject, on the other hand, to a good deal of handling, especially in connection with the Belgian furnace, wherefore their life is short, but it would be equally short if they were made of the most superior quality of refractory material. Generally they will last from eight to 12 days.

Dimensions.—The shape and dimensions of the condensers vary a good

deal according to the method of distillation. In the Silesian practice the condenser is not removed during the maneuver, the residues being withdrawn from the retort through the opening (closed by a tile during the distillation) beneath the condenser and the new charge being thrown in through the condenser, which for that purpose must necessarily be rather large in section. The Silesian condensers are commonly about 1 m. in

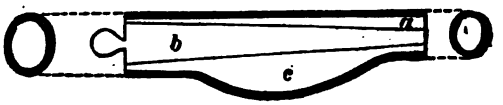
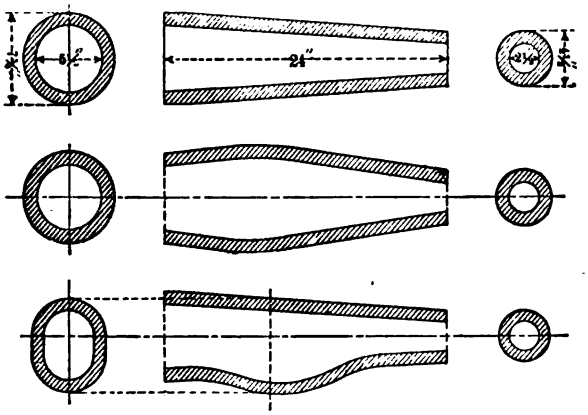


FIG. 181. RHENISH CONDENSER.

length and are prismatic in form. The Rhenish-Westphalian condensers have a belly, as in Figs. 158 and 181, wherein the spelter can collect; in length they vary from 0·7 to 1·1 m. The Belgian condenser is always removed during the discharging and recharging of the retorts. It is made in leg-shape or conical form, the latter being sometimes bulged out with a belly for the spelter to collect in. Various forms of Belgian condensers

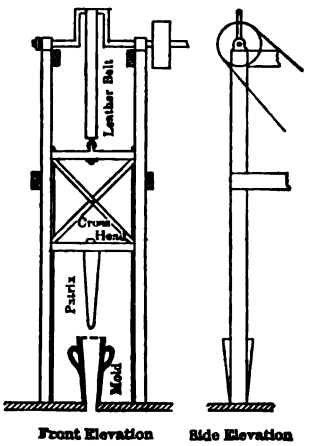


FIGS. 182 TO 184. BELGIAN CONDENSERS.

are illustrated in the accompanying engravings. In the United States the simple conical shape is most commonly employed. The diameter of the Belgian condenser at the inner end is governed of course by the internal dimensions of the retort, since that end of the condenser has to be inserted in the mouth of the retort. At the outer end the condenser must be large enough to admit the insertion of a convenient tool for removal of the liquid zinc. A rather common diameter at the small end is 2¼ in., although

condensers 3 in. in diameter are sometimes found. In length there is considerable variation in the practice. In Pennsylvania the dimensions of the condensers used to be 14×5×2¼ in., all inside measurements; in Kansas, the condensers are generally 18×5½×2¼ in. At Lasalle they are 24 in. long. The Dagner and Kleemann condensers, which are generally used in Upper Silesia, are described in Chapter XII.

Molding.—Belgian condensers are commonly made by hand, a ball of clay being put into a matrix, or mold, of the external shape of the condenser and formed with the aid of a patrix of the internal shape of the



FIGS. 185 AND 186. SKETCH OF MACHINE FOR MAKING CONDENSERS AT LASALLE, ILL.

condenser. One man can make usually about 200 condensers in 10 hours, if they be of the ordinary conical form. In Missouri and Kansas their manufacture is frequently contracted at $1·20 per 100 for labor. When the condensers are made with a belly, a long narrow V is cut out of the edge at the large end, and the two parts being drawn together and united, the desired bulge is formed. This, of course, adds another process to the manufacture and increases the cost. The leg-shape condensers are prepared in a similar manner, a cut being made in each end.

At Lasalle condensers are made with a machine, and so far as I know, that is the only place where that process of manufacture is employed. The machine consists of a plunger of the same shape as the inside of the condenser. It is connected with a cross-head, which works up and down between two vertical guides. A mold is set in position under the plunger

and over a hole in the floor through which the excess of clay finds egress. The plunger is moved up and down by the crank, which is connected with the cross-head by a piece of leather belt. With the aid of one of these machines one man can make 250 condensers in 10 hours, including the finishing work, which has to be done by hand. After they have been molded they are set aside to dry for one day, after which the edges are smoothed by hand, the larger end being set in an iron ring on the floor which is exactly the size of the interior of the retort, so that the condenser will be sure to fit.

Silesian condensers are prepared by folding a sheet of clay of the requisite thickness around a wooden form, the seam being carefully united with the thumbs. The ordinary condenser is said to cost about 50 pfg. (about 12c.), and Dagner condensers 25 pfg. per section.[1] The Dagner condenser, together with the Kleeman, is now most commonly employed in Upper Silesia. According to Georgi,[2] the older form of condensers used to be prepared by the smelters, each smelter making his own supply, even compounding the batch for them; they were dried on the roofs of the distillation furnaces and were employed unburned. The reference to the "older form" of condenser is purely relative, inasmuch as they were preceded by the ancient knee-form condenser (vide Chapter X); the latter were still in use at the Bobrekhütte as late as 1877.[3]

The Rhenish-Westphalian condensers are made in about the same manner as the Silesian, but the form consists of two dovetailed parts, which can be pushed together so that after the condenser is finished it can be easily removed (vide Fig. 181). At Dortmund the condensers are used unburned, being dried on the roof of the distillation furnace and then put directly in place in the niches.[4]

The Belgian condensers are generally stacked in the storeroom of the pottery and when sufficiently dry for safe handling are burned in kilns as required. Before use on the furnace they are commonly whitewashed on the inside to prevent crusts from adhering to them. Sometimes they are glazed inside before burning. Owing to the frequent handling to which they are subjected the life of Belgian condensers is only eight to 12 days. The Silesian condensers last longer. They have to be exchanged when the accretion of zinc oxide crusts inside of them prevents the introduction of the charging scoop, which usually occurs after 14 to 21 days of use, but with rich ore, according to Georgi,[5] may occur in a shorter time.

[1] Th. Beckert and Albano Brand, in O. Dammer's Handbuch der chemischen Technologie, Vol. IV.

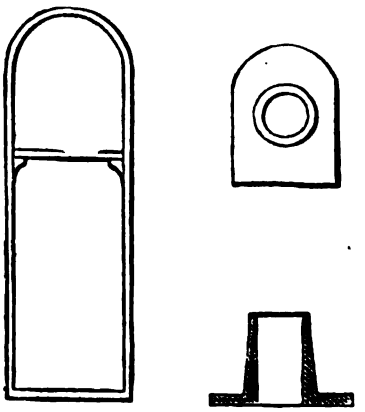
[2] Berg- u. Hüttenm. Ztg., March 2, 1877.

[3] Berg- u. Hüttenm. Ztg., XXXV, 73.

[4] Rev. Univers. des Mines, 1894, XXV, 38.

[5] Loc. cit.

CONNECTIONS.—The condenser is simply luted into the retort with plastic clay, which is stamped with a hot iron of suitable shape so as to make a tight joint. When prolongs are employed, they may be slipped over the outer end of the condenser if the latter be of conical shape, or a small clay tube may be inserted in order to make the connection. The shape of the Silesian and Rhenish condensers generally makes such tubes necessary; they are sometimes as long as 0·2 m. Such a connecting tube is shown in Fig. 158. The Silesian muffle has a step for the condenser to rest on, as shown in Fig. 187. Figs. 188 and 189 represent connections for cylindrical condensers, formerly used. The outer end of the Silesian condenser is closed with an iron plate, having a small hole, which is tightly luted in. The



FIGS. 187 TO 189. CONNECTIONS OF SILESIAN MUFFLE.

connections of the various kinds of retorts are shown in numerous of the illustrations in Chapter X.

DRYING THE RETORTS.—The finished retorts, whether made by hand or by machine, are removed to the drying house, where they remain for a period varying from six weeks to six months, the longer the better. The retorts are first placed in a room at normal temperature until stiff enough to be handled safely, remaining there for a period of 10 to 15 days. This is the critical period in the seasoning of the retort, and especially so are the first two or three days. Sometimes, as at Lasalle, Ill., the retorts are carried to the first drying room in wooden molds, made in segments, which can be taken away so as to leave staves girdled with bands around the retorts which will hold their shape until sufficient water has dried out to remove all danger of distortion. This precaution is advisable especially where the character of the clay or method of manufacture require that a

large percentage of water be used in the batch. Retorts made by hydraulic pressure, however, are usually stiff enough to stand without danger even when freshly made. During the first period of drying care must be taken that no draught of air strike upon the retorts, which might cause them to crack and at any rate would be apt to weaken them. When the retorts have been dried sufficiently to be handled without danger, they are removed to the hot room, where they remain until all of their mechanically contained water is eliminated.[1] The temperature maintained in the hot room is usually about 30° to 35° C. (86° to 95° F.), the hotter the better. Some potteries are arranged so that the retorts may be removed from one hot room to another and perhaps a third, with gradual increase of temperature.

In the storerooms the retorts are stacked upright, open end upward, and commonly as close together as possible. It is advisable, however, to leave a little clear space around each retort so there will be a free circulation of air. In estimating storage capacity for Belgian retorts of the ordinary dimensions it is safest to allow 1 sq. ft. of floor space per retort, besides the necessary alleys for access to the stacks. An area of 100 sq. ft. will afford room for somewhat more than 100 retorts, even if the external diameter of the latter be 11 in., but it is desirable to provide a surplus storage capacity rather than a deficient. The aggregate storage capacity required is calculated from the maximum number of retorts in the furnaces and the length of time they are to be kept before use. Thus if the plant comprises 1344 retorts and the breakage is estimated to be 3% per day and the retorts are to be kept 120 days before using, there must be storage for $1344 \times 0{\cdot}03\ 120 = 4838$ retorts. The daily make of retorts should be stamped with the date on each piece and stacked together.

Condensers do not require so careful handling in drying or so long a time as the retorts. After they have become thoroughly stiff they may be stacked in tiers two or three high for economy of floor space.

Design of the Pottery.—The pottery, that is to say, the division of the works in which the retorts and condensers and perhaps the special fire clay shapes used in the construction and repair of the distillation furnaces are made, comprises two sections: (1) The factory, and (2) the storehouse, which are usually combined in one building. The factory contains the machinery required for the manufacture of the articles above mentioned, while the storehouse is arranged to secure a slow and gradual drying out of the water introduced during their manufacture. The store-

[1] Determinations by Mühlhaeuser at an American works showed 5·16% combined water and 15·88% free water in a freshly made retort. After drying 63 days the percentage of free water was 0·80 to 1·18. These retorts were made of 44 parts raw clay and 56 parts chamotte, ground to pass a No. 10 sieve.

house constitutes a very important part of the pottery, since owing to the long time that the retorts must be kept before use, an extremely large area of floor space is required to keep them.

In designing a pottery, it is advisable to bear four points especially in mind. These are: (1) Safety against destruction by fire; (2) retention of heat in the most complete manner possible; (3) arrangement of a system of heating and ventilation which is adequate and under control; and (4) economical handling of material.

I have set down the safeguarding of the pottery against fire as the point which is of prime importance, since if the stock of retorts be destroyed, there is not only the loss of that stock and the manufacturing plant, but the entire operation of the works is necessarily suspended until a new plant can be built and a fresh lot of retorts has been kept for at least two months. A loss entailed in that manner would be obviously enormous, and the only way in which it might be reduced would be the possibility of borrowing from a neighboring works, which could not be, however, counted upon safely.[1] Although the pottery is not a building which is filled with a large quantity of combustible material and the danger from fire is not very great, there has been to my knowledge at least two cases in the United States within the last few years where the pottery was destroyed by fire communicated from the boilerhouse, which was placed in close proximity to it. In American zinc smelteries the pottery is commonly built close to the steam plant, and frequently in the same building therewith, for the sake of the more convenient transmission of heat and power. These advantages can be preserved, however, without incurring any danger.

In designing a pottery to be fireproof a good form of building would appear to be one which follows somewhat the specifications of the standard New England mill construction, i.e., a building with solid brick external walls, solid floors of heavy plank, grooved and splined, laid on massive timber girders, and a plank roof with tin plate or tar and gravel covering. A construction of the above type, which costs 70@75c. per sq. ft. of floor space, is easily adapted to the requirements of the pottery of a zinc smelting works. The massive brick external walls, solid floors and solid roof would reduce to a minimum the loss of heat by radiation, which is an important consideration in the American climate where a temperature of 30° to 35° C., or more, is to be maintained throughout the year, during a season of which the external temperature may be as low as —25° C. (—13° F.).

[1] The transportation of retorts from one works to another is costly, because of their great weight and the considerable expense of handling and packing, while the loss by breakage may be a serious matter: the transportation of retorts in the winter is especially hazardous because of the danger of their being frosted.

The necessary heat may be imparted to the various rooms by direct radiation from steam pipes, or by indirect radiation, i.e., a preliminary heating of the fresh air by radiating steam pipes and forcing by means of a fan or blower the air so heated to the point where required. If the latter system be adopted, the hot air is best led to various parts of the building through flues contained in the walls or in false pilasters built against them.

In heating the pottery it should be borne in mind that it is not merely necessary to impart the number of heat units to the air of the storerooms to keep them at the required temperature, but there must be a movement of fresh air sufficient to carry off the moisture which is eliminated from the drying clay vessels. If there were no such system of ventilation, the air would become saturated with moisture and the drying would cease, although the temperature were maintained at a high point. This would be especially the case in a dryhouse of the type indicated above, in which the construction is designed not only to avoid loss of heat by radiation but also by leakage.

Where fuel is so cheap as it is in the smelting districts of the United States ordinarily no special attention is directed toward economy in the drying house. The house is commonly built of brick, but the walls are seldom so thick as they should be and the loss by leakage is apt to be large. Ventilation is generally haphazard. The heating is effected in the United States usually by steam; in Europe hot air furnaces are frequently employed. The floor on which the retorts are set is sometimes laid with close joints; sometimes with open spaces which permit the hot air to rise from the lower rooms. In the United States the potteries are built two or three stories high; in Kansas they are generally only two stories.

Improvements in the pottery storerooms tend not only to economize heat, that is to say, fuel and money, but also to hasten the drying of the retorts, reducing the length of time which they must be kept for thorough seasoning; and what is still more important, making a more evenly dry and better retort, which may lead to economy in the subsequent distillation.

Glazing Retorts.—At some zinc smelteries the retorts are treated with a glaze before insertion in the distillation furnace, receiving a coat either outside or inside, or both. Such glazes are fusible mixtures, which during the annealing of the retort will frit and make a more or less smooth and dense surface, which should tend to prevent the escape of zinc vapor from the retort; for example, there are works in Europe which use a glaze made up of 60 parts clay, 30 parts of pulverized glass, and 10 parts of soda, these proportions being by weight. Common salt, alone, is sometimes employed; also a solution of water glass, or a mixture of water glass and clay. At Pittsburg, Kan., a glaze composed of 50 parts alum, 40 parts pulverized

glass and 10 parts soda, has been used experimentally, though the practice of glazing the retorts is by no means general in that district.

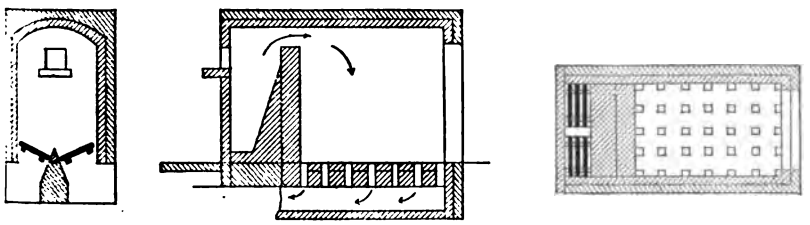
With many kinds of clay the retorts acquire naturally a glaze in the distillation furnace, the fine particles of ash carried over from the fireplaces by the draught collecting on the outside of the retorts and forming a fusible coating, which possibly to a certain extent prevents the escape of zinc vapor. Whether the latter will find egress into the combustion chamber depends obviously upon the difference in the tension of the vapor inside of the retorts and that of the gases of combustion on the outside. If that difference is sufficient, the vapor will overcome the resistance of any glaze that is likely to remain upon the exterior surface of the retorts. The formation of a natural glaze takes place especially with fuels which have a high percentage of iron in the ash, as is the case with the coal used in the western part of the United States. At works in Illinois, Missouri and Kansas the retorts very soon become covered with a dense brownish-black glass, and the corrosion is so active that considerable quantities of slag drip off from the retorts. That of course hastens the destruction of the retorts. The action takes place with furnaces heated by producer gas as well as those heated directly from a grate. With the former type of furnaces, which have a solid hearth, it is necessary from time to time to remove the stalagmites which accumulate on the hearth. It is noteworthy that in the furnaces fired with natural gas there is no such formation of a slag glaze, the fuel in this case being free from ash.

ANNEALING FURNACES.—After the retorts have thoroughly dried in the storeroom of the pottery, they must be annealed or tempered, because it would never do to put these comparatively fragile vessels of fire clay directly into the distillation furnaces where the temperature is at least at red heat. They must, therefore, be first placed in a special furnace in which they can be burned gradually. In this manner the water of constitution is slowly eliminated and after burning the retorts are transferred still hot to the distillation furnace. This refers of course only to the daily replacement of defective retorts. In starting a furnace on a fresh campaign, with a full quota of new retorts, the gradual heating of the furnace permits the retorts to be installed directly. Condensers are burned in the same manner as retorts, but they are permitted to cool before using, since they are not suddenly exposed to intense heat when put in place in the distillation furnace. In Upper Silesia, Rhenish Prussia and Westphalia condensers are sometimes put in use without a previous burning, as has been remarked in a foregoing section of this chapter.

In Europe the annealing of the retorts is often done in ovens connected

with the distillation furnaces, the escaping gases of combustion from the latter being utilized therein.[1] Drawings of several furnaces, especially of the Silesian and Rhenish types, with such ovens will be found in Chapter X. Inasmuch as such ovens are generally small, containing only a few retorts, it is customary to provide an independent annealing furnace of larger size, common to a group of distillation furnaces, which is kept full of hot retorts as a reserve in case of the requirement of more than the oven combined with the distillation furnace will furnish. At other works the entire supply of retorts is annealed in independent furnaces, which is the universal practice in the United States.

The annealing furnaces are very similar to the kilns employed for burning any kind of refractory ware, but are of smaller dimensions than those used in the pottery industry. They are essentially small boxes of fire



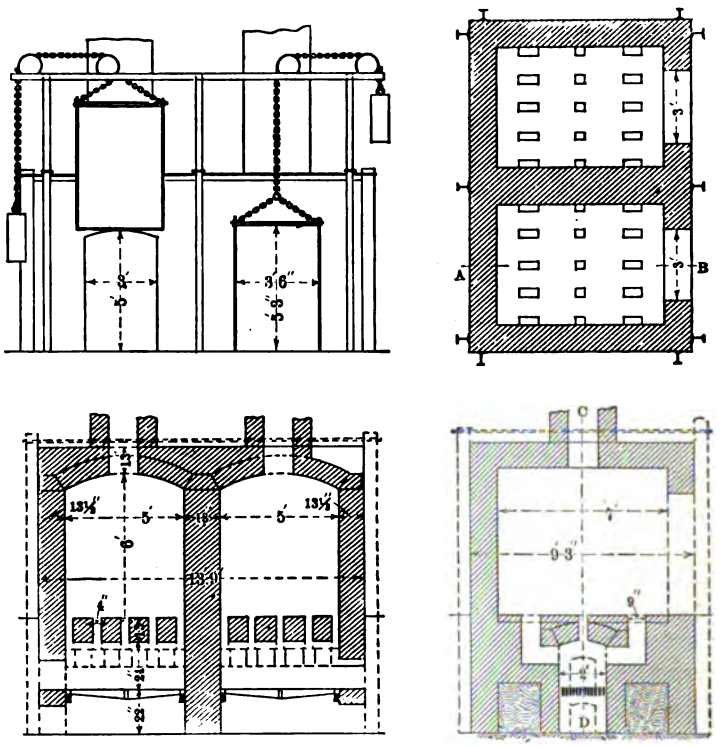
FIGS. 190 TO 192. DOWN-DRAUGHT ANNEALING FURNACE.
Fig. 190: Transverse section. Fig. 191: Longitudinal section. Fig. 192: Plan.

brick, with an arched roof and an open floor of fire clay tile, forming a kind of grating, upon which the retorts are placed, standing on end. Annealing furnaces are of two types, namely the up-draught and the down-draught. In the former the fireplace is arranged under the perforated hearth and the flames pass upward through the holes in the latter, between the retorts, and then to the chimney; in the latter type the fireplace is arranged at one side of the furnace and the flames pass upward over a fire bridge into the top of the furnace chamber, whence they draw downward between the retorts and escape through the holes in the hearth into a flue which leads to the chimney. The down-draught type of kiln is generally preferred. Kilns are built either single or double, two being united side by side.

The general proportions of a double furnace of each type are shown in the accompanying engravings. The capacity of such a furnace is governed, of

[1] The annealing of retorts in that manner is comparatively free from the objection to the utilization of the waste heat of the distillation furnaces for blende roasting, because the annealing process is itself an intermittent one, of which the rise and fall may correspond somewhat to those of the process of distillation.

course, by its hearth area and the external cross-sectional dimensions of the retorts. The single furnace which used to be employed at Pittsburg, Kan., was commonly of a size to hold about 30 retorts, one furnace usually supplying the whole plant. At Iola they build smaller furnaces and put one between every pair of distillation furnaces (blocks). These kilns being heated by natural gas are quite simple, the gas burners being introduced

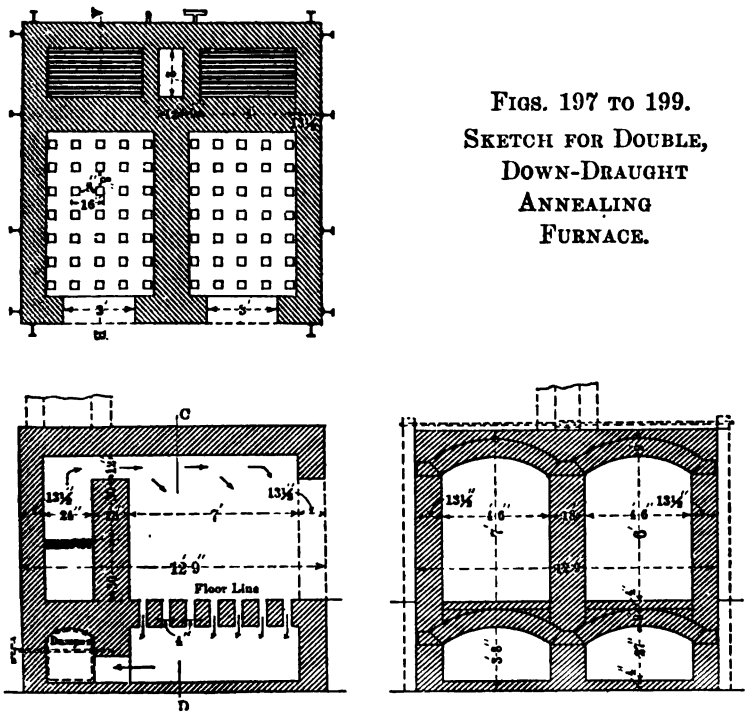


FIGS. 193 TO 196. SKETCH FOR DOUBLE, UP-DRAUGHT ANNEALING FURNACE.

through ports on opposite sides directly under the grated tiling on which the retorts stand.

The number of retorts that must be burned daily depends upon the average breakage in the distillation furnaces. Since, however, in giving any average, there will be some days when more are broken than on other days, preparations must be made to supply the maximum number. The requisite number of retorts is transferred from the storehouse to the kilns

during the morning of each day. The kilns are fired up about noon, gently at first and gradually with more coal, until finally a temperature of approximately 750° to 800° C. is attained. The maneuver on the distillation furnaces begins about 6 a. m., and lasts until 10 or 11 a. m., during which any defective retorts are replaced, new retorts being taken from the tempering furnace as required. This is done by three men, one of whom reaches into the furnace with a pair of tongs and grasping a retort by an edge of the open end drags it out of the furnace. An iron bar, usually a piece of 1 in.



FIGS. 197 TO 199.
SKETCH FOR DOUBLE, DOWN-DRAUGHT ANNEALING FURNACE.

or 1¼ in. pipe, 10 or 12 ft. long, is then inserted in the retort while two men pass a bent bar under the retort and thereby carry it to the distillation furnace, the third man maintaining the balance by means of the long bar. Arrived at the distillation furnace, the two men holding the bent bar, raise the end of the retort to the shelf of the niche in which it is to be inserted, and the third man shoves it into place by means of the long bar. After the tempering furnace has been discharged it is allowed to cool down, and when it is sufficiently cool to permit access to its interior a fresh lot of

retorts, the supply for the next day, is introduced. The actual time of tempering is generally about 15 hours, but it may be as short as 12 hours. It should be sufficiently long to insure the elimination of the water of constitution of the clay, which is expelled between 375° and 660° C., after which the retort is in safest condition for handling. Condensers are burned in large lots and kept in stock, which is drawn upon as required.

It should go without saying that the annealing kilns ought to be situated conveniently with reference to the distillation furnaces, which they are to serve. It is not difficult to arrange them so when the distillation furnaces are large and require many retorts, but with small furnaces it is not so easy. For example, a plant comprising five Belgian furnaces of 224 retorts each, a total of 1120 retorts, with a maximum breakage of 3% will require only 34 retorts per day, a number that can be annealed in a comparatively small kiln, which may be made to serve the five distillation furnaces, although if the latter are arranged in line, as is the common way, Nos. 1 and 5 will be rather remote from the tempering furnace. This difficulty is illustrated in many of the works of Kansas and Missouri. At Iola, where the distillation furnaces are larger, it is the general practice to place a tempering furnace between two opposite distillation furnaces, whereby each kiln serves 600 or 660 retorts.

NON-CORROSIVE RETORT LINING.—Benjamin Sadtler, of Denver, Colo., has patented[1] a zinc retort which consists of the ordinary cylindrical vessel of refractory material, provided with a basic lining about 0·125 in. in thickness. As basic material, magnesia, chromite, corundum, or titaniferous iron ore, may be used; no special claim is made to any particular substance. This lining is prepared by application of a solution of sodium silicate to the interior of the retort, after which the refractory material pulverized to pass a 20-mesh sieve, or about that size, and free from fine dust is introduced, and by rotation of the retort in a suitable framework is made to adhere to the inner walls as an even coating. When the retort is burned in the ordinary manner the sodium silicate sinters with the clay of the retort on one hand and the coating of refractory material on the other and produces a firm and dense lining, which is claimed to resist the corrosive influence of high percentages of iron, manganese, etc., which would be fatal to distillation in an ordinary clay retort. By the use of the retorts above described the inventor proposes to treat the mixed sulphide ores of Colorado and elsewhere by roasting and distillation, and patents have also been taken out on that process.

[1] United States Patent No. 642,722, Feb. 6, 1900; No. 656,268; Aug. 21, 1900; British Patent Nos. 1892, 1893 and 1894, all of Jan. 30, 1900.

A furnace to demonstrate the utility of the Sadtler retorts was erected at Denver and ran for a brief campaign, as to which Professor Sadtler communicated the following results:[1]

"Two separate runs were made, one of 89 consecutive days, the other of 42. The furnace contained 30 retorts of the regular size, which were purchased in Missouri. They were lined with various basic substances. In the run of 89 days a loss of 10 retorts was experienced, all of which was by breakage, none being corroded. The total loss was equivalent to a daily loss of one in 267. The ore which was smelted assayed before roasting from 16 to 30·5% Fe, from 1·5 to 12·89% Pb, from 2 to 9% SiO_2 and from 16·5 to 37·5% Zn; its silver content varied between 5 and 17 oz. per 2000 lb.; its gold content from $1 to $11 per 2000 lb. The spelter which was produced assayed from 0·52 to 1·12% Pb and from 0·06 to 0·12% Fe; it was free from gold and silver. The residues drawn from the retorts assayed from 4·2 to 8·5% Zn and contained all of the gold, silver, copper and iron originally present in the ore and practically all of the lead. In the second run of 42 days the residues assayed only 5·4% Zn. The recovery of zinc ranged from 70 to 84% and was irregular because the temperature of the small furnace could not be governed properly. The ores treated were not roasted so low as 1% S, but notwithstanding that the percentage of zinc retained by the residues from the retorts was very low. This was attributed to reduction of metallic iron, which decomposed the imperfectly roasted blende and liberated its zinc. A good deal of iron shot and some large pieces of metallic iron were found in the retort residues. This iron contained a noteworthy quantity of the precious metals, especially gold."

The Sadtler retorts were subsequently used in regular operation by the Midland Smelting Co., of Bruce, Kan., where ferruginous blende from Leadville, Colo., was treated successfully, it is reported. The argentiferous residuum drawn from the retorts was sold to lead smelters. A difficulty was experienced through cracking of the non-corrosive lining, which thereby afforded the slag an opportunity to eat through the retort. The slag dripping down upon the underlying retorts of the same tier, the latter were corroded from the outside. To obviate that difficulty the plan of giving the retorts an external coating on the side which is to lie uppermost in the furnace has been proposed. Since the works at Bruce were closed, they having had direct, coal-fired furnaces, which are no longer economical in Kansas, the Sadtler retorts have been adopted by the Cherokee-Lanyon Spelter Co. at its works in the Iola district, where all the furnaces have been provided and run with them for more than a year. Crushed chromite is

[1] Eng. and Min. Journ., Sept. 29, 1900.

employed as the lining material. The cost of the latter is the chief element of expense in preparing the retorts. According to Professor Sadtler,[1] one man at $2 per day with a helper at $1·50 can line 70 or 80 retorts, making the labor cost approximately 4·67c. The material required per retort is 10 to 12 lb. of crushed chrome ore, assaying 50% Cr_2O_3, and 2·5 lb. of silicate of soda. The cost per retort may be reckoned therefore as, labor, 4·67c.; 11 lb. chrome ore @ 1·5c., 16·5c.; 2·5 lb. silicate of soda @ 1·2c., 3c.; total, approximately, 25c.

[1] Private communication.

VIII.

FUEL AND SYSTEMS OF COMBUSTION.

The zinc distillation furnace is essentially a simple heating furnace, i.e., nothing is required of the combustion of the fuel beyond the production of the necessary temperature and no particular chemical conditions such as an oxidizing atmosphere (vide the roasting furnace) or a neutral atmosphere (vide the refining furnace) have to be maintained. The distillation furnace is therefore analogous to the pot melting furnace for glass making and even more analogous to the retort furnace used for the distillation of coal for illuminating gas. The chief part of the zinc distillation furnace is therefore to present a laboratory in which a high temperature may be attained and maintained in an economical and practicable manner. Other considerations are the building of a strong and stable structure and its arrangement for the proper reception of the retorts. Before taking up the design of such furnaces, however, it is well to review the principles of the combustion of fuel, whereby the required heat is obtained.

The combustion of fuel is purely a chemical operation, wherein oxygen is made to combine with some other element with which it has an exothermic, or heat evolving, reaction. Considered thus broadly many elements are fuels, but in general metallurgy only carbon, hydrogen and sulphur are of great importance in the development of heat by combination with oxygen, and commonly under the classification of fuel we refer only to carbon in its various forms and the combinations of carbon and hydrogen. Those substances are utilized as bituminous and anthracite coal, lignite, peat, coke, wood, charcoal, petroleum and natural gas, which vary in composition and character as elsewhere described. In their combustion the carbon and hydrogen of their constitution combine with oxygen in definite proportions, expressed by chemical formulæ and liberate a definite amount of heat measured in arbitrary units.

Quantitative Measurement of Heat.—The quantitative measurement of heat is referred to water and the conventional thermometric scales as

standards. Thus the calorie is the quantity of heat required to raise 1 kg. of pure water one degree centigrade in temperature at or near 4° C.; this being sometimes an inconveniently large unit the gram-calorie is frequently used, which is the quantity of heat required to raise 1 g. of water 1° C. in temperature. The British thermal unit, or B. T. U., is the quantity of heat required to raise 1 lb. of water 1° F. at or near 39·1° F., while the pound-calorie is the quantity required to raise 1 lb. of water 1° C. Consequently 1 calorie=3·96832 B. T. U., and 1 B. T. U.=0·251996 calorie; 1 pound-calorie=2·2046 B. T. U.=0·55556 calorie. The B. T. U. is equivalent to 778 ft.-lb. of energy, and 1 h. p. being equivalent to 33,000 ft.-lb. per min. corresponds to 42·416 B. T. U. per min.

The conversion of the quantity of heat developed by the combustion of 1 kg. of a substance expressed in calories into the equivalent of 1 lb. expressed in British thermal units may be done as follows: One calorie =3·968 B. T. U.; but since 1 kg.=2·2046 lb., 3·968÷2·2046=1·8. Therefore, if the number of calories generated by 1 kg. be multiplied by 1·8 the product will be the number of B. T. U. per lb. For example, if the heat of combustion of 1 kg. of C to CO_2 be 8080 calories, that of 1 lb. is 8080 ×1·8=14,544 B. T. U.[1] Similarly, since 1 B. T. U.=0·252 calorie the conversion of B. T. U. per lb. into calories per kilogram is made by multiplying by 0·55555, which factor is obtained from 0·252×2·2046= 0·55555.

The quantities of energy and heat given in thermal equations refer to such quantities of the substances as amount to their formula weight in grams. For example, the thermal equivalent of carbon burned to CO_2 is 96.960 calories. In the equation $C+2O=CO_2$, 12 g. of carbon (its equivalent weight) are used. The unit of carbon is 8080; hence 8080×12= 96,960 gram-calories=96·960 kg.-calories.

Classification of Coals.—The only kinds of fuel which concern the zinc smelter, at present at least, are coal, either anthracite or bituminous, and natural gas. Of the two, coal is by far the more important, it being the fuel upon which the European smelters and the Eastern smelters of the United States are obliged to rely; it is only the smelters of Indiana and Kansas who enjoy the luxury of natural gas.

The coals available for fuel are classified variously, especially with respect to the bituminous coals. An old classification based solely on the character of the coke produced, divided coals into two classes, caking or non-caking, according as the coke produced formed a compact or pulveru-

[1] This is the determination of Favre and Silberman. Berthelot gives 14,647. The figure 14,600 is frequently employed.

lent mass. Percy classed coal into three varieties: (1) Non-caking, or free-burning, rich in oxygen; (2) caking; (3) non-caking, rich in carbon. This classification was based on the chemical composition of the coals and therefore on their calorific power. Another classification, much used in Europe, divided coal not only according to the character of the residue left after dry distillation, but also according to the length of the flame produced in combustion. Thus Gruner distinguished five types of bituminous coal, as follows:[1]

(1) Non-caking coals with long flames: These coals which most closely approach lignite in character yield 55 to 60% of pulverulent coke, the evolution of volatile matter giving rise to a long smoky flame. In composition they show 75 to 80% C, 4·5 to 5·5% H and 15 to 19·5% O and N, the ratio of the oxygen to the hydrogen being 3:1 or 4:1.[2] In calorific power they range from 8000 to 8500 calories, 1 lb. of coal being capable of evaporating from 8 to 10 lb. of water.

(2) Caking, long flame gas coal: The coals of this type yield 60 to 68% of caked, but very friable and porous, coke and 32 to 40% of volatile matter, of which 17 to 20% is gas. In composition they vary from 80 to 85% C, 5 to 5·8% H and 10 to 14·2% O and N, the calorific power ranging from 8500 to 8800 and the factor of evaporation from 8 to 9·7.

(3) Bituminous or furnace coal: These coals burn with a smoky flame, at the same time softening and intumescing in the fire. They yield 68 to 74% of caked and swollen coke and 15 to 16% of gas. In composition they contain from 84 to 89% C, 5 to 5·5% H and 5·5 to 11% O and N, the ratio of the oxygen to the hydrogen being 1:1. Their calorific power varies from 8800 to 9300 and their factor of evaporation from 9·7 to 11.

(4) Caking coals with short flame: These yield 74 to 82% of caked and very compact coke and 12 to 15% of gas. They contain 88 to 91% C, 4·5 to 5·5 H and 5·5 to 6·5% O and N, the ratio of the oxygen to the hydrogen being 1:1. Their calorific power varies from 9300 to 9600 and the factor of evaporation from 11 to 12.

(5) Anthracite coals: These yield 82 to 92% of pulverulent or fritted coke and 12 to 8% of gas. They burn with a short flame. They contain from 90 to 93%C, 4 to 4·5% H and 3 to 5·5% O and N, the ratio of the

[1] Roberts-Austen. Introduction to the Study of Metallurgy. fourth American edition. p. 214.

[2] The percentage of nitrogen in coal is rarely more than 1·5%, and generally is only about 1%. The percentage of oxygen in a coal has an important effect on its heating power. It is not known in precisely what form it exists, but it is not free and is assumed to be present in combination with hydrogen, the availability of which for further oxidation is thus reduced by one eighth of the weight of the oxygen, wherefore "available hydrogen"=H - 0·125 O.

oxygen to the hydrogen being 0·5:1. Their calorific power varies from 9200 to 9500 and their evaporative factor from 10·8 to 11·4. These coals are intermediate between the bituminous coals and the true anthracite of the United States (vide p. 261).

The reason why some coals should have the caking property and others should not is not clear, non-caking coals being often of very similar chemical composition to those in which the caking property is highly developed. The caking coals undergo an incipient fusion or softening when heated, so that the fragments coalesce and yield a compact coke, while the non-caking coals (also called free-burning) preserve their form, producing a coke which is serviceable only when made from large pieces of coal, the smaller pieces being incoherent. It is found that caking coals lose their property when exposed to the air for a long period or by heating to about 300° C., and that the dust or slack of a non-caking coal may in some cases be converted into a coherent coke by exposing it suddenly to a very high temperature.

The distinction between long-flaming and short-flaming coals is not often made in the United States. A long-flaming coal is simply one having a high percentage of volatile matter, which gives off a long flame when burned in an ordinary furnace because of the difficulty of supplying the volatile matter with a sufficient quantity of hot air to insure its complete combustion. The manner in which coal is burned has a great effect upon the flame. Charcoal, for example, if burned with free access of air merely glows; but if burned with a limited supply of air in a thick bed, wherein the products of combustion from the lower part will pass through the upper part, carbon monoxide will be formed and will burn with a blue flame. The same phenomenon can be produced in the case of hard coal and bituminous coal low in volatile matter, and is taken advantage of in the method of clinker grate firing, wherein a thick bed of coal is carried on the grate to effect an incomplete combustion in the fireplace followed by a secondary combustion of the carbon monoxide in the furnace, thus elongating the flame of a lean coal. This verges upon gas firing.

Coals are also classed sometimes as "lean" or "fat," which classification corresponds more or less to "short-flaming" and "long-flaming." All of the above classifications are more common in Europe than in America.

A convenient classification of coal is based on the relative percentages of fixed carbon and volatile matter contained in their combustible portion determined by proximate analysis. Such a classification as is commonly employed in the United States is shown in the following table[1]:

[1]William Kent, Steam Boiler Economy, p. 42.

Class.	% Fixed carbon.	% Volatile matter.	Heating value per lb. of combustible B. T. U.	Relative value of combustible.
Anthracite.....	97·0 to 92·5	3·0 to 7·5	{14,600 {14,800	93
Semi-anthracite....	92·5 " 87·5	7·5 " 12·5	{14,700 {15,000	94
Semi-bituminous....	87·5 " 75·0	12·5 " 25·0	{15,500 {16,000	100
Eastern bituminous.	75·0 " 60·0	25·0 " 40·0	{14,800 {15,200	95
Western bituminous.	65·0 " 50·0	35·0 " 50·0	{13,500 {14,800	90
Lignite.....	under 50	over 50	{11,000 {13,500	77

PRINCIPLES OF COMBUSTION.—The combustion of coal or natural gas is simply a chemical process, whereby their carbon and hydrogen constituents are oxidized to carbon dioxide, carbon monoxide and water, a definite quantity of heat being developed by the respective reactions. The quantity of heat developed is the same whether a pound of coal be burned in one minute by rapid combustion in a furnace, or in ten years by slow oxidation under exposure to the atmosphere, but the temperature attainable depends upon the rapidity of the combustion, or rather upon the means of conserving the heat evolved. The theoretical temperature attainable in the combustion of any fuel may be calculated from the following formula:

$$T=(B.T.U.)\div(W\times S),$$

in which B. T. U. = number of British thermal units generated by the combustion; W=weight of the gaseous products; and S=specific heat of the gaseous products.

It is evident from the above formula that the rapidity of the combustion is not a function of the temperature, which is determined by the conservation of the heat evolved. Inasmuch, however, as the quantity of heat radiated from a given mass of fuel is a function of the time, a larger proportion of the heat generated may be lost by radiation when the combustion is very slow, wherefore the temperature that can be obtained is indirectly determined by the rate of combustion, and in general the more rapid the combustion the higher the temperature that can be produced. The advantage of high rates of combustion is commonly overestimated, however, since at an ordinary rate of 10 lb. of coal per square foot of grate surface per hour in fire brick furnaces the percentage of heat lost by radiation is small and, according to William Kent, the actual temperature that may be attained will be very nearly as high with that rate of combustion as with a rate of 20 or 40 lb. of coal per square foot per hour. Practically the tem-

perature of a fire is largely a function of the proportion of excess air used in the combustion of the fuel.

The supply of air also governs, together with other conditions, the degree of combustion of coal and natural gas, as well as of other fuel. In the case of coal, when the combustion is complete, the whole of the carbon is burned to dioxide, CO_2, each pound of carbon generating 14,600 B. T. U.; and the whole of the hydrogen is burned to aqueous vapor, H_2O, each pound of hydrogen generating 62,000 B. T. U. This development of heat is not fully realized, however, since a part of that resulting from the combustion of the hydrogen is absorbed as the latent heat of evaporation of the 9 lb. of water produced per pound of hydrogen, while another part is consumed in superheating to the temperature of the furnace that steam and also the steam that may be derived from moisture in the coal or in the air supplied to the furnace.

The ability to attain a high temperature at a comparatively moderate rate of combustion must not be confused with the development of the requisite quantity of heat, which is a direct function of the quantity of fuel burned. Furnaces in which a high temperature and great quantity of heat are both necessary are generally designed therefore for a high rate of combustion on the grate.

Imperfect Combustion.—Imperfect combustion results when the carbon of a fuel is converted into monoxide, CO, instead of into dioxide, CO_2; the formation of carbon monoxide may result either from the direct oxidation of carbon to that product, or from the reduction of carbon dioxide by another molecule of carbon according to the equation:

$$CO_2+C=2CO.$$

The above reaction takes place when the carbon dioxide produced by the combustion of carbon on the grate is reduced in passing through a bed of red-hot coke by another part of carbon. This is a cooling process, in which 10,150 B. T. U. are absorbed per pound of carbon originally burned to dioxide, wherefore if the reduction occur to the extent that all the dioxide is reduced to monoxide, the heat generated by the combustion of 1 lb. of carbon is $14,600-10,150=4450$ B. T. U. This reaction and its thermal results are very important considerations in gas firing.

Imperfect combustion results also from the distillation of the hydrocarbons, or volatile matter, of coal which may escape unburned, or of which only the hydrogen may be burned leaving the carbon in the form of soot or smoke to be carried off in the gases passing out of the furnace. All the products of imperfect combustion, whether it be carbon monoxide or hydro-

carbons or soot produced by the dissociation of the latter may be burned, however, if they be brought into contact with a sufficient supply of highly heated air in a very hot chamber. This is contrary to the statement to be found in many treatises on the combustion of fuel, which has been copied from one of the old works, to the effect that smoke once produced cannot be burned.[1] William Kent, a well known and expert authority on this subject, demonstrated the fallacy of that statement by a simple experiment. A short piece of candle was placed inside of a tall narrow tin cylinder. The deficient supply of air the candle thus received caused it to give off a column of black smoke. This was made to pass into the central draught tube of a Rochester kerosene lamp. As it passed up into the flame of the lamp it was completely burned, not a trace of smoke being visible in the lamp chimney. The experiment was also made with a larger volume of smoke, produced by burning paper under the lamp, with the same result.[2]

CALCULATIONS RESPECTING THE COMBUSTION OF FUEL.—The ultimate analysis of a fuel being known, i.e., its percentage of carbon, hydrogen, sulphur, nitrogen, etc., the weight and volume of the air required for its combustion, and the weight and volume of the gases that will be produced, can be calculated just as in the case of any chemical reaction and by the same rules. In designing a furnace for metallurgical purposes in which the combustion of fuel is so highly an important matter as it is in the distillation of zinc ore, it is evident that in order to obtain the maximum efficiency the proportions of the furnace, including the grate area, the volume of the laboratory or combustion chamber, and the area of the flues and chimney and the height of the last should be planned with reference to the volume of the gases that must pass through them, their temperature and other factors. However, this is but rarely done, not merely in the design of zinc-smelting furnaces, but in all other kinds of metallurgical furnaces, and indeed the subject has been as yet studied so imperfectly by metallurgists that much of the data that is required for such calculations is still lacking. The design of metallurgical furnaces in accordance with well known physical laws has not yet been attempted, except in few instances, and naturally little is to be found with respect thereto in existing metallurgical treatises. In so far as gas firing is concerned, however, an excellent presentation has been made in a treatise on *Regenerativ-Gasoefen*, published in 1898, by Friedrich Toldt, an instructor at the K. K. Bergakademie at Leoben, Austria, whose work may be studied profitably.

Data as to the quantities of material involved in the combustion of vari-

[1] Charles Wye Williams, on the Combustion of Coal and the Prevention of Smoke.
[2] Steam Boiler Economy, p. 9.

ous fuels are given in the following table, which is transcribed from the treatise by Toldt referred to above:

QUANTITY OF AIR REQUIRED FOR THE COMBUSTION OF ONE KILOGRAM OF VARIOUS FUELS AND WEIGHT AND VOLUME OF GASES PRODUCED.

Fuel	Burned to	Weight and Volume.	Require			Produce				
			O	N	Air	CO	CO_2	H_2O	N	Total
C	CO	Kg.	1·33	4·46	5·79	2·33			4·46	6·79
		Cu. m.	0·93	3·55	4·48	1·86	....	.....	3·55	5·41
CO	CO_2	Kg.	0·57	1·91	2·48		1·57		1·91	3·48
		Cu. m.	0·39	1·53	1·92	....	0·80	.....	1·52	2·32
C	CO_2	Kg.	2·67	8·93	11·60		3·67		8·93	12·60
		Cu. m.	1·87	7·15	9·02	....	1·87	.....	7·12	8·99
H	H_2O	Kg.	8·00	26·72	34·72			9·00	26·72	35·72
		Cu. m.	5·59	21·29	26·88	....	....	11·19	21·29	32·48
CH_4	CO_2+2H_2O	Kg.	4·00	13·36	17·36		2·75	2·25	13·36	18·36
		Cu. m.	2·80	10·64	13·44	....	1·40	2·79	10·65	14·84
C_2H_4	$2CO_2+2H_2O$	Kg.	3·43	11·49	14·92		3·14	1·29	11·49	15·92
		Cu. m.	2·40	9·12	11·52	....	1·60	1·60	9·12	12·32

One kilogram of oxygen unites with 3·33 kg. N to form 4·33 kg. of air. One liter of oxygen weighs 1·430 g.; 1 l. N=1·255 g.; 1 l. air=1·294 g. These weights are computed at 0° C., 760 mm. atmospheric pressure and the latitude of Berlin (Landolt and Börnstein, Physikalisch-chemische Tabellen, p. 77).

WEIGHT IN POUNDS OF AIR REQUIRED FOR THE COMBUSTION OF CARBON, HYDROGEN, HYDROCARBONS AND SULPHUR.

Fuel	Burned to	Lb. O	Lb. N	Lb. air	Lb. gas produced
C	CO_2	2·67	8·85	11·52	12·52
C	CO	1·33	4·43	5·76	6·76
CO	CO_2	0·57	1·90	2·47	3·47
H	H_2O	8·00	26·56	34·56	35·56
CH_4	CO_2+2H_2O	4·00	13·28	17·28	18·38
S	SO_2	1·00	3·33	4·33	5·33

Quantity of Air Required Practically for the Combustion of Coal.—Theoretically the combustion of 1 lb. of carbon to dioxide requires 11·52 lb. of air. Practically under the ordinary conditions of chimney draught that quantity is greatly exceeded. Donkin and Kennedy showed in the results of 16 tests with steam-boiler installations that the air supply ranged from 16·1 to 40·7 lb.[1] The effects of an excess of air upon the combustion of coal are to reduce the temperature produced thereby and increase the relative weight of the products of combustion. Although the initial volume increases with the excess, however, it is to be noted that the relative volume just after passing through the fire remains practically constant because

[1] Walter R. Snow, "The Influence of Mechanical Draft upon the Ultimate Efficiency of Steam Boilers," a lecture delivered before the Engineering Society of Columbia University, December 1, 1898.

of its lower temperature and consequently greater density.[1] In so far as the temperature is reduced there is a loss of efficiency, since the lower the initial temperature the less rapidly will the gases of combustion transmit their heat, and the final result is that, within practical limits, the temperature of the escaping gases is highest with the greatest excess of air supplied.

In burning 1 lb. of carbon to dioxide there are generated 14,600 B. T. U. The products of combustion comprise 3·667 lb. of carbon dioxide and 8·853 lb. of nitrogen, the total weight being 12·52 lb. Assuming the specific heat of carbon dioxide to be 0·217, and that of nitrogen to be 0·2438, the average specific heat of the gas is 0·2359. According to the formula given in a previous section the theoretical elevation of temperature of the fire above the atmospheric temperature would be $14{,}600 \div 12{\cdot}52 \times 0{\cdot}2359 = 4942{\cdot}5°$ F. (2728° C.). If the atmospheric temperature were 62° F., the theoretical temperature of the fire would be $4956° + 62° = 5004{\cdot}5°$. It is probable that the specific heat of gases of combustion at high temperatures is higher than 0·2359, which would have the effect of reducing the temperature. The actual specific heat of combustion of the gases under those conditions has not been determined, but the figure of 0·237 is commonly assumed in temperature calculations. However, because of the excess of air required to effect complete combustion, besides other considerations, it is never possible to attain the theoretical temperature. The effect of different percentages of air supply in reducing the temperature of fire is shown in the subjoined tables, which are taken from Kent's treatise on *Steam Boiler Economy*, wherein they are credited to H. T. De Puy, of the Babcock & Wilcox Co.:

CARBON BURNED TO CO_2 WITH EXCESS OF AIR.

(Heat value of carbon assumed to be 14,600 B. T. U. and specific heat of gases 0·24. Temperatures are expressed in degrees Fahrenheit.)

Air supply above 11·52 lbs., per cent.	25	50	75	100	150	200
Air per pound of carbon, lbs.	11·40	17·28	20·16	23·04	28·80	34·56
Products of combustion, lbs.	15·40	18·28	21·16	24·04	29·80	35·56
Elevation of temperature of fire	3950°	3328°	2875°	2530°	2041°	1711°

CARBON BURNED PARTLY CO_2 AND PARTLY TO CO, WITH EXCESS OF AIR.

Carbon burned to CO_2, per cent.	100	80	60	40	20	0
" " " CO, "	0	20	40	60	80	100
Excess of air, per cent.	50	40	30	20	10	0
Products of combustion, lbs.	18·28	15·52	12·99	10·67	8·60	6·76
Elevation of temperature of fire	3328°	3375°	3350°	3323°	3139°	2743°

In order to realize the theoretical temperature of combustion it would be necessary that the air should be delivered to the incandescent fuel at a

[1] Snow, loc. cit.

perfectly uniform rate; that the combustion should be complete with the theoretical quantity of air required; and that there should be no loss by radiation from the incandescent fuel into the surrounding furnace walls. These conditions can be more nearly obtained in the combustion of gaseous fuel than in the combustion of solid fuel, and that is one of the important advantages of gas firing, as will be pointed out more fully in a subsequent section of this treatise. Even with gaseous fuel, however, an excess of air is required for complete combustion, the percentage being the less the higher the air is preheated. According to H. H. Campbell[1] with gas at 600° C. and air at 50° C., from 20 to 100% of air in excess is necessary to prevent the escape of an appreciable quantity of unburned combustible; with the air and gas at 1000° C., however, 10% is a large excess, while 5% is sufficient for ordinary producer gas. With dust fuel burned with an intimate and regular admixture of air in a chamber having thick brick walls complete combustion may also be attained with a small excess of air.

With insufficient air supply the actual temperature is always less than the theoretical, for the reason that some of the oxygen passes through the fire without combining with carbon. Generally the air supply is not regular, even with steady draught pressure, for the reason that the freshly fired coal chokes to some extent the air passages through the bed of fuel on the grate, causing the formation of some carbon monoxide and chilling the furnace. William Kent states that he has obtained temperatures exceeding 1650° C., as measured by a Uehling & Steinbart recording pneumatic pyrometer, with Pittsburg (Pa.) coal containing less than 2% of moisture and having a calorific power of 15,000 B. T. U. per pound of dry combustible, other conditions being a fire brick combustion chamber and frequent firing of small quantities of coal at a time; this corresponds nearly to the theoretical temperature due to the air supply of 19 lb. per pound of combustible, which is the figure found in practice to give the highest efficiency of steam boiler performance.[2]

It is sometimes possible for a high percentage of carbon monoxide and a great excess of air supply to exist at the same time, which is explained by the supposition that the excess of carbon monoxide be generated at one portion of the grate surface and that the excess of air enter at another, and that the two currents, one of carbon monoxide and the other of air, be never brought into contact until their temperature is reduced below the point of ignition.

Ignition Temperature of Fuels.—With any fuel a certain initial temperature must be imparted to it before combustion will take place, i.e., the

[1] Trans. Am. Inst. of Min. Eng., XIX, 151. [2] Steam Boiler Economy, p. 31.

rapid combustion that is required in metallurgical practice; slow combustion, or gradual oxidation may take place at ordinary temperatures. The ignition temperature of fuels varies according to their physical properties and chemical composition. Extreme density in solid fuel and great tenuity in gaseous fuel reduce the inflammability; for the former reason anthracite coal is more difficult to ignite than bituminous coal, while to start combustion of graphite a still higher temperature is required. Fuels richest in hydrogen are the easiest to light, the inflammability of resinous wood and cannel coal being due to that fact. Gases will not, however, ignite below a red heat, notwithstanding a high tenor in hydrogen, because of the extreme dispersion of their molecules. Pine wood ignites at 295° C.; ordinary bituminous coal at 325° C., while coke, anthracite, hydrogen and carbonic monoxide require a dull red heat.[1] Conversely when burning substances are cooled below their ignition temperature the fire is extinguished; i.e., chemical action ceases.

FLAME.—Flame is a mass of intensely heated combustible gas, containing incandescent particles of solid matter. It is not necessarily gas in a state of combustion, for combustion cannot take place without access of air and flame may exist, as in passing through a furnace and flue, where there is no supply of air to burn the gas. If the flame in passing through a tube becomes cooled below a bright red heat, the gas will not burn when it escapes and comes into contact with cool air, because its temperature will be reduced below its ignition point; it will be simply chilled and pass off unburned, a condition which is usually, although not necessarily, manifested as smoke.

Visible flame is an evidence of imperfect combustion or non-combustion, the product of the perfect combustion of carbon being carbon dioxide gas, which is invisible, and that of hydrogen being aqueous vapor, which is also invisible. Mr. Kent, who has aptly pointed out how the principles of combustion may be learned from simple experiments, illustrates this with the example of the central draught kerosene lamp. The wick being properly adjusted, the lamp gives a rather short and clear white light without a trace of smoke. If without altering the adjustment of the wick the opening at the bottom of the central draught tube be gradually obstructed, the flame grows longer and its whiteness changes to yellow and then to red; it begins to smoke, and finally when the supply of air is nearly shut off the flame has risen nearly to the top of the chimney and a dense column of black smoke and soot is given off. This experiment shows that with the same consumption of fuel, i.e., the oil supplied by the wick, the flame may be

[1] Roberts-Austen, Introduction to the Study of Metallurgy, p. 171.

short and intensely hot; or very long, of a low temperature, smoky and sooty. While the flame is lengthening and before it becomes smoky the combustion may be complete, but it is not effected in as short a space as it was with the original supply of air. For a given supply of fuel a short flame means rapid and complete combustion, a long flame delayed combustion and a very long flame imperfect combustion. If midway in the flame of medium length a cool surface be interposed, the temperature of the flame will be lowered, the combustion will be rendered imperfect and smoke and soot will be produced.[1]

EFFECT OF ASH AND MOISTURE IN COAL.—In speaking of the calorific power of coal reference is made generally to the number of heat units that will be generated by the burning of its combustible constituents. Besides this, however, all coals contain a percentage of ash and moisture, both of which detract from the actual calorific power, not only because ash and moisture are not combustible, but also because of other results which ensue from their presence. In other words the quality of a coal is not exactly indicated by the calorific power of its combustible matter. If the coal be high in ash not only is its heating power reduced in direct proportion to the percentage of ash, but also the quantity of ash formed on the grate tends to check the air supply and therefore to diminish the rate of combustion. If the coal be also high in pyrites (sulphur) the ash is likely to fuse into a clinker, the silica combining with the oxide of iron produced by the burning of the pyrites, and thus may choke the grate completely, necessitating frequent cleaning of the fire. If the coal be high in moisture, or in oxygen, not only will the heat units derived from a pound of it be low, but the attainable temperature will also be lower than that which a better coal will yield. In order to develop the requisite temperature with poor coal high in ash it is necessary to have either a larger grate surface or stronger draught than with good coal. Sometimes the clinkering of the ash is so troublesome that strong draught is of no avail, and in such cases large grate surface is absolutely required. When the ash in a coal is so large in amount as to necessitate frequent cleaning of the fire its effect in reducing the value of the coal is greater than what is due to its mere percentage, because of the loss in unconsumed coal to which it leads (vide page 287).

DIRECT FIRING.

The combustion of coal is effected practically in three ways, namely: (1) Burning directly on a grate, which is referred to as "direct firing"; (2) burning in such a way as to produce a combustible gas, which is then

[1] Steam Boiler Economy, p. 9.

burned secondarily in the laboratory of the furnace—known as "gas firing"; and (3) burning directly in the form of dust in the laboratory of the furnace. Direct firing and gas firing merge into each other in such a way that it is difficult to draw a sharp dividing line between them. The following sections refer to the ordinary direct firing of anthracite and bituminous coal.

BURNING OF ANTHRACITE COAL.—The burning of anthracite coal for the purpose of zinc smelting is far less important than the burning of bituminous coal, since it is a kind of fuel which is available only to the smelters of one district, who produce a comparatively small quantity of spelter. The practical combustion of anthracite is free from the difficulties which attend the combustion of bituminous coal and no great degree of skill is required to fire with it, especially when the large sizes are used. All that the stoker needs to do is to keep a level bed of coal on the grate of a depth proportionate to the strength of the draught, to watch carefully to prevent the formation of air-holes in the bed of incandescent fuel and to clean the fire at intervals of six to 10 hours. With ample draught the stoker can regulate the combustion by varying the thickness of the bed of coal or altering the position of the damper.

With a given draught, if the bed of coal be too thin an excessive supply of air passes through it, reducing the temperature of the fire and causing a waste of heat in the chimney gases; if it be too thick, some of the carbon will be burned only to monoxide instead of to dioxide, also causing a reduction of temperature and loss of heat. The latter trouble is manifested by a sluggish action of the fire, the presence of blue flames on the bed of coal and the low temperature of the furnace. The remedy is either to carry a thinner bed of fire or to open the damper and produce a stronger draught, in other words to give the fire a greater supply of air. Bad firing because of an excess of air is more common than because of a deficiency and is not so clearly manifested. It is due especially to carrying too thin a bed of fuel and may be overcome by increasing the thickness of the latter just short of the point where blue flames make their appearance. This condition being observed the highest temperature is attained by firing small quantities of coal at a time.

Small Sizes.—The burning of small sizes of anthracite is more difficult than the burning of the large sizes, because the fineness of the coal tends to choke the air passages through the bed of fuel on the grate, wherefore a thinner bed has to be carried unless there is a very strong draught and a thin bed is more difficult to keep free of air-holes than a thick one. The small sizes of anthracite are usually higher in ash than the large sizes, and the

fire must therefore be cleaned oftener, an operation which always chills the fire and causes a waste of heat. For the combustion of the fine sizes of anthracite it is common to use a forced blast under the grate, furnished either by a fan or by a steam jet, preferably the former, which both practice and experiment have shown to be the more efficient. The distillation furnaces fired with anthracite small coal which are used in New Jersey and Pennsylvania are commonly provided with an under-grate blast. In the case of coal which is high in pyrites, however, and is consequently liable to clinker badly and rapidly destroy the grates, the steam jet may be the more advantageous. In burning fine anthracite coal there is always a greater loss of fuel passing through the grate unburned than in burning the larger sizes, although grates with more or less proportionately small openings are used. Experience has also shown that the combustion of small anthracite is more perfect when the coal remains undisturbed, or as nearly as possible in the condition in which it was put into the fire, than when it is turned over so that the partly consumed and partly unconsumed coal are mixed together.

Burning of Bituminous Coal.—The great difficulty in effecting economical combustion of bituminous coal is with respect to the hydrocarbons and carbon monoxide which are likely to escape unburned. In the ordinary method of firing with bituminous coal there is an intensely hot bed of coke lying upon the grate bars. When it is necessary to replenish the fire, which in rapid driving will generally be required at intervals of five or 10 minutes, a lot of fresh coal, much of it of fine size, is spread evenly over the bed. While the furnace door is open there is a great influx of cold air, reducing the temperature of the furnace as a preliminary to the uneconomical cycle of phenomena which is to ensue. The first effect of the addition of fresh coal is to choke the air spaces existing in the bed of coke, thus shutting off the supply of air which is needed to burn the gases produced from the fresh coal. Next there is a rapid evaporation of moisture from the coal, which absorbs heat and thereby robs the furnace. The aqueous vapor which is formed may perhaps be reduced by carbon to hydrogen and carbon monoxide, a reaction which also absorbs heat. If the hydrogen and carbon monoxide were burned again an equivalent quantity of heat would be regenerated and there would no loss, but ordinarily there is not enough air in the furnace to combine with them and they escape unburned. There is also a deficiency of air for the combustion of the hydrocarbons and carbon monoxide which are subsequently developed.

It is a frequently adopted expedient to introduce air into the combustion chamber above the fire to supply the oxygen necessary to complete the burn-

ing of these gases, and experience shows that an advantage may be gained if the quantity of air admitted be small and judiciously regulated; but the great volumes of black smoke which issue from such furnaces even under those conditions indicates that the combustion is still far from complete. The failure to complete the combustion in that manner is due to imperfect admixture of the air and reduction of the temperature below the ignition point of the gas. As is well known, one of the quickest ways to cool off a furnace is to open the fire door and let in a great volume of cold air. In order to effect a complete combustion by the admission of secondary air it is necessary that the air be highly preheated and be allowed sufficient time and space to mix thoroughly with the gas. We have an example of this kind of firing in the Boetius and other systems of semi-gas firing.

After the moisture has been expelled from the coal the distillation of the hydrocarbons begins and a considerable portion of them escapes unburned unless they be supplied with the necessary quantity of air under the proper conditions as referred to in the previous paragraph. These gases being comparatively cool cannot be burned unless the air for the secondary combustion be highly heated. After a while the distillation of the hydrocarbons proceeds at a slower rate, the fine coal which at first obstructed the supply of primary air through the grate is partially burned away, sufficient hot air begins to come through the bed of hot coke to burn thoroughly all the gases and the combustion is then for a brief period effected economically as is manifested by the chimney gases becoming smokeless or nearly so, whereas a few minutes previous great volumes of dense smoke were being poured forth. If the fire were then to be left alone it would burn economically until the air spaces became so large as to admit an excessive supply of air, when there would be a loss of heat on that account. Before that time is reached, however, in ordinary firing fresh coal is thrown on the fire with the same sequence of results as outlined above.

Requisites for Complete Combustion.—The requisites for the complete combustion of bituminous coal are an adequate supply of air, a sufficient space and length of time for the gases to burn and a thorough mixture of the air and gas at the point where ignition is taking place, that is at the place where the temperature is hot enough to cause combustion. Mr. Kent illustrates the importance of effecting an intimate mixture by supposing a can of kerosene oil to be ignited in the open air; it will not burn with a clear white flame as in an Argand burner nor with a transparent blue flame as in a blow-pipe, but will produce a dark reddish flame giving off dense clouds of smoke. In this case there is plenty of time, space and air available to effect complete combustion, but an intimate mixture of the com-

bustible gas and the ample supply of air is lacking. If on the other hand the oil be burned in a lamp and fed slowly upward through the wick, the oil itself or the vapor distilled from the top of the wick will be brought into contact with air which has been heated to a very high temperature by radiation from the flame and perfect combustion will ensue. Should the wick be turned a trifle too high, the lamp will smoke, for although there is plenty of air present, it is not hot enough to cause combustion.[1] This example illustrates another important point in practical metallurgy, i.e., that the transference of heat from what is generated by the combustion of a fuel to the air necessary for the combustion of a subsequent portion of the fuel may be both advantageous and economical; in fact, it is what takes place in every direct combustion. This is an entirely distinct principle from that of heat recuperation, because the heat imparted to the air is not recovered from what would be otherwise wasted, but is taken directly from that which is generated by the combustion and then is restored thereto. This principle is applied in many systems of semi-gas firing, in roasting furnaces and in other metallurgical furnaces. The preheating of the air by recuperation of heat from the waste gases is the ultimate step in the economical combustion of fuel. From a theoretical standpoint the ideal method of burning coal is first to convert it into gas in a producer and then to burn that gas without losing any of the heat of the primary combustion in a fire brick chamber with thick walls, the air for the secondary combustion being highly preheated by recuperation from the chimney gases.

Improved Methods.—The improved methods of effecting complete combustion and preventing smoke which are applicable in steam boiler practice cannot be generally adopted in metallurgical work, because of the difference in the dimensions of the grate surfaces. On the other hand some of the arrangements which in steam boiler practice have to be made by special design, in metallurgical practice coincide directly with the requirements of the furnace. The fire box of a steam boiler is provided commonly with a broad and comparatively short grate. In metallurgical furnaces the grate surfaces are likely to be narrow and comparatively long; especially is the latter the case in the conventional type of direct-fired Belgian zinc-distillation furnace.

Coking System of Firing.—This depends upon the principle that the whole bed of fire shall not be covered with fresh coal at the same time. It is applied in steam boiler practice by throwing the fresh coal only on the front half of the fire, leaving the rear half with a level bed of partly burned coal. The gases distilled from the fresh coal then pass over the rear half,

[1] The Mineral Industry, VIII, 132.

through which an excess of air is entering, said air being heated as it passes through the incandescent coke. The two currents of gas, one containing the distillation products and the other the supply of hot air, commingle in the combustion chamber. When nearly all the gas has been distilled from the pile of coal in the front half of the furnace, the pile is pushed and leveled over the rear half, and either immediately or within a minute or two, according to whether the gases have been more or less completely driven off, fresh coal is again piled in front. This system involves a greater amount of labor and attention on the part of the stokers than ordinary firing, and its advantage depends upon the skill with which it is applied.

Alternate Firing.—Another method of firing which has all the advantages of the coking system, without the disadvantages of the latter, is that known as alternate firing, which consists in throwing fresh coal first on one half of the bed of fire and then on the other half, alternately at equal intervals of time. Alternate firing is of no use, however, unless there is a large combustion chamber in which the two gaseous currents are mixed and allowed to burn.

Special Furnaces and Mechanical Stokers.—The special furnaces for use in connection with steam boilers, like that of William Kent, which aim to insure an intimate admixture of the gases in a space sufficiently large for their complete combustion, only accomplish that which is to be found in many metallurgical furnaces, including reverberatory smelting furnaces and zinc distillation furnaces. The automatic stokers which have been used advantageously in steam boiler practice have also been applied with excellent results to reverberatory roasting furnaces. Such installations may be seen at the Pueblo and Kansas City plants of the American Smelting & Refining Co., where the well known American stoker is used. Cases have been reported in which the ore has failed to burn in furnaces provided with these stokers, while the fire-box has been rapidly destroyed. The explanation is too perfect combustion, the gases from the fireplace not having sufficient excess of air to oxidize the ore. The remedy is the admission of the requisite supply of additional air.

Clinker Grate Firing.—In metallurgical practice the depth of the bed of fuel on the grate is of much importance not only with respect to economy in the combustion, but also with respect to other results. In cases where the coal is lean and short-flaming and the heat radiated directly from the fire is insufficient to do the work that is desired it is necessary to employ a deep bed of fuel, i.e., the so-called clinker grate firing. In this the fire box is virtually a gas generator, a deep layer of clinkers being maintained upon the grate bars. This layer of clinkers serves to heat the air passing through

and enables it to unite above the fire with the gas generated from the imperfect combustion of the coal. The flame of an otherwise lean coal is thereby elongated. This method of firing is commonly employed with the Belgian type of distillation furnace used in the St. Louis district and at Pittsburg and vicinity in Kansas, where the furnaces are designed so that the top of the bed of fuel will be from 3 to 4 ft. above the surface of the grate bars. The fire boxes are long and narrow, being only 16 in. wide at the level of the grate. The grate itself consists merely of two bars of iron, 2×2 in. in cross-section, resting upon two bearing bars. Resting on the grate there is a mass of clinker which supports the incandescent fuel. The coal which is burned is rated as slack, but comprises pieces up to nut size.

In general it may be considered that with clinker grate firing there will be less loss through unconsumed coal in the ash than where the fire itself is kept down to the level of the grate bars. At most of the copper smelteries in the United States, however, the deep clinker grates have been discarded. A bituminous or semi-bituminous coal which gives a sufficiently long flame is used, and experience has shown that for their conditions, where fast driving of the furnace prevails, a thin bed of fuel with live coals nearly down to the grate bars gives the best results, even if attended with considerable waste of fuel. If fuel be expensive and economy in its use important the fragments of cinder or coked coal that drop through the grates at the frequent barrings may be cheaply recovered by washing. In the reverberatory copper smelting furnaces of the United States it is common to carry a bed of coal 24 in. deep, but Peters states that the most rapid smelting is done with only 19 in. of depth, the coal containing 10% ash.[1]

Grates and Grate Bars.—Grates for metallurgical furnaces are arranged in three forms: (1) horizontal plane; (2) sloping plane; and (3) step. The last type is commonly installed as a combination of step and sloping plane. Of the three types the horizontal plane and step grates are the most commonly used in metallurgical furnaces; the inclined plane grate finds extensive employment in steam boiler practice. With good coal the horizontal grate is the best, permitting the maintenance of an even bed of fuel (which is essential to good firing) to a degree which is not possible with any other form. The inclined grates which are used under steam boilers are given only a slight slope; installed in that manner they do not interfere materially with proper regulation of the fuel bed and afford easy access to the under side of their extreme end, which is an advantage when the ash pit is necessarily long and shallow. Step grates are employed especially with inferior coal, being capable of arrangement with

[1] Modern Copper Smelting, seventh edition, p. 460.

large air spaces, without permitting the small coal to run through. It is difficult, if not impossible, to maintain an even bed of fuel upon them and on that account modern gas producer practice has pronounced against them. They were a prominent feature in the original Siemens gas producers, in which the idea was that the fuel dumped in from the top would form a natural talus corresponding to the slope of the grate, but such was not the case.

Open and Closed Ash Pits.—Equally important to the type of grate is the type of fire box, which may be fired either from the sides or from the end and may have either a closed ash pit or an open one. In metallurgical furnaces the common types are closed fire boxes, with fire and ash pit doors on the sides; and open fire boxes, following the Welsh practice, in which the fire is cleaned from the back, though the coal is generally thrown in through doors in the sides; with the latter system the ash pit may be still closed in, but it is not so convenient to arrange the doors as the other way. The only advantage of the Welsh system is the facility with which access is had to the fire and the ash pit, and the ability to use very short grate bars which are easily cleaned and removed. There is, however, generally an open space above the grate, if the latter be horizontal, through which air makes a short circuit, which is precisely what ought to be avoided, and even under the best conditions the air supply cannot be so well regulated as with the other type of fire box, which is in practice decidedly the more economical. The foregoing remarks apply especially to reverberatory furnaces. The direct fired Belgian distillation furnace belongs typically to the closed ash pit class, though the special conditions of its installation cause but little attention to be given to the regulation of the air supply by the ash pit doors, which if they exist at all have to close in the entire tunnel and are seldom capable of permitting a close regulation of the air supply.

Forms of Grate Bars.—In either horizontal or inclined grates two kinds of bars may be used, namely the simple bar of wrought iron, square in section, laid loosely on the bearers, or the cast iron bar of the conventional fish-belly form, so arranged as to fit tightly in the fireplace. For horizontal grates in roasting furnaces the cast iron fish-belly form is the best adapted, the spaces between the bars being fixed uniformly, which tends toward economy in the combustion of the fuel and also economy in maintenance, the bars being the deepest at the middle, which is the point where they are weakest and most liable to destruction. Moreover, with cast iron bars the proportion of free air space in the grate can generally be made much larger than with loose wrought iron bars, which can seldom be used less than one inch wide. The rectangular form is the best for step grates, although

the durability is less, but deviations from the rectangular form for the sake of durability do not tend to even stoking with this type of grate.

The conventional cast iron grate bar is tapered in cross-section so as to make a wider opening between the bars at the lower edge than at the upper edge, and thus insure the free passage of the ash which drops between them. Projections are cast on the sides of the bars to keep them at the proper distance apart. The width of the bars and of the air spaces between them is determined according to the kind of fuel that is to be burned. For fine sizes of anthracite bars ¾ in. wide with ⅜ in. openings are commonly used. For pea or nut coal ½ in. bars and ½ in. openings. With run of mine bituminous coal cast iron grate bars are commonly made ½ in. wide, with spaces of ⅝ in., ¾ in. or even 1 in. between them. The total area of the air spaces is therefore generally 50%, more or less, of the total area of the grate surface, but with the clinker grates used in the Western distillation furnaces of the Belgian type there is an example of 2 in. bars with 4 in. spaces, the latter constituting 75% of the total grate area.

In any kind of firing when the ash of the coal tends to clinker narrow air spaces are objectionable, because they are liable to becomc clogged and are difficult to keep open so as to allow a sufficient supply of air to pass through them. The resistance to the passage of air through the grate and the bed of coal lying upon it depends upon the size of the coal as well as the dimensions of the air spaces, besides the quality of the coal as regards coking or non-coking property, the thickness of the bed of coal and ashes, the presence or absence of clinker, etc. Coals that are low in ash, the latter being non-clinkering, can be burned on grates with very narrow air spaces. Providing the proper rate of combustion can be maintained, the smaller the air spaces the greater will be the economy of fuel, because less will be lost by falling through the grate unburned. Fine sizes of anthracite coal are sometimes burned on flat cast iron plates perforated with tapering holes about 0·5 in. in diameter at the upper surface, the total air space being about 25% of the grate area. Grates of that type are employed at the zinc smelteries in New Jersey and Pennsylvania, where anthracite coal as small as No. 2 buckwheat is burned.

Gas Firing.

Gas firing broadly considered includes the combustion of all kinds of gaseous fuel, which may be obtained either as a natural product or as one made artificially from coal, wood or other carbonaceous matter. The latter is known most commonly as "producer gas," because it is made in a type of

furnace referred to specifically as a "gas producer"; producer gas is also designated by the terms "fuel gas," "air gas" and "Siemens gas." In metallurgy it is the most important gaseous fuel that we know. Natural gas is available only in a few favored localities, and water gas and coal (illuminating) gas are not used at all; at least not outside of metallurgical laboratories.

Producer Gas.

So far producer gas firing has proved to be the most economical method of burning coal for technological purposes, save for the generation of steam in which line for various reasons it has made no headway. There are numerous comprehensive treatises dealing with this subject, which is so important as to have a literature of its own. The most recent of them are *Regenerativ-Gasoefen* (1898) by Friedrich Toldt and *Le Chauffage Industriel et les Fours à Gaz* (1898), by Emilio Damour, both of which are to be recommended for reference. In this treatise the idea is merely to explain the general principles of gas firing and describe the most important producers which are now employed.

In taking up the subject of gas firing for metallurgical purposes at this time, it is unnecessary to go into the history of the art. It is sufficient to say that the first practicable gas producer is credited generally to Bischof whose furnace was in use as early as 1839. Modern gas firing is due, however, chiefly to the inventions of Frederick and Sir William Siemens, of whose patents the first is dated 1856, but reference is generally made to the celebrated patent of 1861. Producer gas is frequently referred to, even at the present time, as Siemens' gas. Without detracting from the credit due to these and other practical introducers of the art, however, there is no doubt that the rudiments of gas firing are to be found in many of the early furnaces when coal first began to be used as fuel. Thus, as Damour points out, the early Welsh metallurgists in using their domestic anthracite to heat moderately long reverberatory furnaces were accustomed to burn the coal in a bed from 0·75 to 0·80 m. in thickness, thereby producing carbon monoxide, as a method of elongating the short flame of their lean coal. The Welsh clinker grate is essentially a gas producer. Similarly there are many metallurgical furnaces in use at present wherein the direct firing on the grate, which forms a part of the furnace itself, is a species of semi-gas firing, and there is really no sharp dividing line between that method of firing and the producers of the Boetius type or between the latter and the modern gas producers which are shaft furnaces, separate from the furnaces in which their gas is to be employed for heating.

THEORY OF FUEL GAS PRODUCTION.—According to the modern understanding of gas firing, the process consists in the generation of the gas in a special furnace, known as the producer or generator, its conveyance through a proper flue to the place where it is to be burned, and finally its combustion in the special metallurgical furnace which is to be heated by it. The plant consists, therefore, of four essential parts, namely the gas producer, the conduit, the laboratory and the chimney; in modern practice there is generally a fifth part, namely the steam blower or fan, the function of which is to force the combustion of fuel in the producer, while if steam be used it plays also a part in the chemical reactions which take place in the process of combustion. In gas producing installations, where the air is blown into the generator below the zone of combustion, the draught producing power required of the chimney is of course lessened and the chief functions of the latter become in that case the dispersion of the products of combustion formed in the laboratory of the furnace and the introduction of the secondary air. The generator, laboratory and chimney may be arranged at considerable spaces apart, or they may be arranged in close combination. Many examples may be cited of each type of installation.

The gas producer, as distinguished from the simple fire box in which the entire process of combustion is effected, is charged with a bed of fuel so deep that the sucked-in or blown-in air, in predetermined volume, leads to a peculiar process of distillation and combustion. The lowest part of the bed of fuel, which is in contact with the grate, or more properly speaking, the hearth, because all producers do not have grates, is ignited and acts as the agent for carrying out the process. It is immaterial, theoretically, if there be a bed of ashes immediately above the grate, or hearth, and below the zone of combustion, though practically the ash zone may serve a useful purpose. The fuel charged through the top of the producer passes in its descent through three stages: (1) drying; (2) distillation; and (3) combustion. The air entering at the bottom of the producer burns to ash the coke which exists at that level, forming carbon dioxide. The carbon dioxide thus produced, together with the nitrogen and unconsumed oxygen of the air, rises through the bed of incandescent fuel, wherein higher up the free oxygen is consumed and all or nearly all the carbon dioxide is reduced to carbon monoxide according to the formula:

$$CO_2+C=2CO.$$

The heat of combustion of carbon when it has burned to carbon dioxide is 97 kilogram-calories. Carbon monoxide when burned to carbon dioxide disengages 68 calories. The total heat disengaged being the same, whether

carbon is burned directly to dioxide or if it be burned first to monoxide and then to dioxide, it follows that the heat of formation of carbon monoxide is 29 calories. Expressed differently, 1 lb. of carbon burned to 2·33 lb. of carbon monoxide develops 4450 British thermal units, or about 30% of the total carbon energy; in the secondary combustion, 2·33 lb. of carbon monoxide burned with oxygen produce 3·66 lb. of carbonic acid and develop 10,150 heat units, or 70% of the total energy; making in all 14,600 heat units for the complete combustion of the original pound of carbon. It is evident, therefore, that if the heat of the primary combustion of carbon to monoxide be not employed in some useful manner, 30% of the energy of the fuel will be theoretically lost; i.e., the gas will carry into the furnace only 70% of the total energy of the carbon. In practice there are further losses, as will be discussed in a subsequent section, while on the other hand part of the heat of combustion of C to CO is utilized in the dissociation of steam, which heat is subsequently restored to a large extent when the gas is burned, and the loss of the sensible heat of the primary combustion may be further minimized by placing the producer close to the furnace.

In the zone of the fuel bed above that in which the reduction of carbon dioxide takes place, the gas should no longer contain theoretically any oxygen, although practically there will be a small percentage. The chemical action which takes place in this zone is a species of dry distillation, similar to that which occurs in the retorts employed for the manufacture of illuminating gas and in by-product coke ovens, whereby there are formed heavy carbureted hydrogen, CH or C_4H_4, together with various analogous compounds, such as propyls, butyls, amyls, etc.[1] Further heating decomposes the heavy carburets into carbon and light carbureted hydrogen, according to the equation:

$$C_4H_4=2C+C_2H_4 \text{ (ethylene)}.$$

Other products richer in hydrogen and free hydrogen itself are formed in the same manner as the decompositions take place in every process of dry distillation. The producer gas at that point, therefore, consists chiefly of carbon monoxide, nitrogen, heavy and light carbureted hydrogen and free hydrogen, together with small quantities of unreduced carbon dioxide and water. In the uppermost part of the fuel bed, where the temperature is lowest, the formation takes place of the distillation products, generically known as coal tar, and the moisture of the fuel is evaporated.

Steinmann says:[2] "The above reactions begin at red heat with fuels in general, but proceed in a different manner with different fuels, with the

[1] Steinmann, Compendium der Gasfeuerung, p. 2. [2] Op. cit., p. 4.

result that the gas produced from stone coals has the highest temperature, while wood gas has usually the lowest. In general these temperatures depend upon the depth of the fuel bed on the one hand and on the quantity of chemically combined or mechanically associated water on the other hand. The presence of water in the fuel has always a cooling effect, and since the water content of wood, if the latter has not been brought to a minimum by sharp drying, is the highest of all fuels, its gas has the lowest temperature. Next to wood gas come the gases of air-dried peat and lignites.

"Ethyl gas (C_4H_4), which is the heaviest constituent of generator gas, is set free in the greater quantity the drier the fuel; the regulation of the supply of air and its distribution under the grate is also of importance in this connection. An excess of air or its imperfect distribution favors the formation of carbon dioxide, while a deficiency of air impairs the ethyl gas.

"The oxidation of carbon is a complicated process. It begins at the moderately low temperature of 400° C., carbon dioxide being formed then as the chief product, whether the supply of air be large or small, and only a very little carbon monoxide being formed therewith. The oxidation becomes more active if the temperature rises to 700° C., but the chief product is still carbon dioxide, although the air supply be deficient. Even under that circumstance, which in so far as the proportion of air to carbon is concerned is favorable to the formation of carbon monoxide, only traces of the latter are formed. Above 700° the proportion of carbon monoxide to carbon dioxide increases rapidly until 995° is reached where the former gas is formed exclusively. An increase of the incandescent bed of coal does not suffice to form carbon monoxide if the minimum of temperature (700° C.) be not exceeded. These observations explain why if carbon be oxidized at a lower temperature than 700°, it burns without flame, while if it be oxidized at a higher temperature the combustion is accompanied by a flame. In the first case the carbon burns directly to dioxide, an incombustible gas, and in the latter to monoxide which at a higher temperature burns with a further part of oxygen producing the characteristic blue flame."

It used to be thought that the direct combustion of carbon to monoxide did not take place, it being supposed that the formation of monoxide was due solely to reduction of carbon dioxide previously formed from the carbon.[1] R. Ernst, however, proved the contrary by experiment.[2] He found that the composition of the gaseous products of combustion depends

[1] Vide Gas Producers, p. 115, by F. J. Rowan, a pamphlet reprinted from the Proceedings of the Institution of Civil Engineers LXXXIV, ii, 1885-1886, where the author said "The direct combustion of solid carbon to CO with disengagement of heat, no CO_2 having been primarily formed, was an unknown chemical process."

[2] Journ. Prakt. Chem., XLVIII, 31 to 45.

almost entirely upon the temperature. The formation of carbon monoxide and dioxide begins at about 400° C., the production of the latter increasing rapidly up to 700° C., when it forms about 20% of the gas, while the proportion of monoxide at that temperature is only about 3% of the gas. With further rise of temperature the percentage of monoxide increases at the expense of the dioxide, until at 1000° C. it forms one third of the gas, the remainder being nitrogen from the air. At that temperature carbon monoxide is the direct product of the combustion.

A consideration of these various reactions is important in connection with fuel gas production. Generalizing briefly, O. Boudouard, who is an eminent authority on this subject, has stated that the conditions controlling the reduction of carbon dioxide to monoxide by carbon, as occurs in the gas producer, point to the following requisites in order to obtain the best results: A high temperature, inasmuch as the reduction of carbon dioxide by carbon, which begins at about 550° C., is only complete at 1000° C.; a porous fuel in a finely divided state; and a very slow stream of air and gas.[1]

REACTIONS WITH STEAM.—When the producer is blown with steam, which is now the general practice in Great Britain and America, another reaction takes place in the zone of combustion, namely the decomposition of water by carbon with the formation of carbon monoxide and hydrogen, the reactions which take place in the combustion zone being expressed in this case as follows:

$$C+2O=CO_2$$
$$CO_2+C=2CO$$
$$H_2O+C=CO+2H.$$

Advantages from the Use of Steam.—The effect of the decomposition of water in the zone of combustion is to increase greatly the percentage of hydrogen in the producer gas and consequently its calorific power. This does not of course imply a gain in heat—which would be contrary to the law of the conservation of energy. The calorific power of the gas is increased, but a corresponding quantity of heat is consumed, in the zone of combustion in the producer, in decomposing the water, of which the hydrogen is subsequently burned in the laboratory of the furnace. There is therefore only a restoration of energy, which is probably never complete. The effect of the steam in the combustion zone of the producer is cooling; the combustion of the hydrogen in the laboratory of the furnace affords an increased heat there. It has been previously pointed out that if coal were

[1] Bull. Soc., Chim., XXV, 833 to 840. Important references as to the theory of fuel gas production are Boudouard's paper in Ann. de Chimie et de Physique (7), XXIV; and Bodländer's paper in Zts. f. Elektrochemie, VIII, xliv, 833 to 843, Oct. 30, 1902.

converted simply into gas, which should be allowed to cool before being burned further, there would be a complete loss of the heat of primary combustion, which would amount to 30%, more or less, of the original energy of the fuel. With the use of steam a large part of the heat of primary combustion is used up, at the point where it is developed, in effecting a chemical reaction from which it can be recovered subsequently. In other words, a portion of the heat which might otherwise be wasted is carried to a place where it is useful. There may be therefore an important practical gain from the use of steam, although there is no gain in the total energy over what existed originally in the fuel.

It must be conceded, however, that the use of steam leads to the production of a richer gas, aside from its tenor in hydrogen, inasmuch as a part of the carbon being burned with the oxygen derived from water there is a corresponding reduction in the quantity of diluting nitrogen in the gas, and consequently a saving in heat. Nevertheless it must not be forgotten that coal and labor are consumed in raising the steam to blow the producer; the coal may possibly be saved by developing the steam from the waste heat of the furnaces, but even then there remains the labor and interest and amortization charges on the plant; as a general thing a battery of gas producers is accompanied by a direct fired boiler plant.

Relative Quantity of Steam to be Used.—The quantity of steam that may be used in the production of fuel gas and the pressure under which it is applied vary according to the conditions. In the manufacture of water gas, wherein steam and air are used alternately, the latter is blown in until the fuel bed has attained the proper temperature, after which steam is introduced until the temperature has been reduced to the point where the decomposition of steam is no longer economical. The production of steam fuel gas on the other hand is a continuous operation and a balance must be arrived at between the air and steam which will permit it to go on regularly in the most efficient manner.

Messrs. R. D. Wood & Co.[1] in their pamphlet on *Fuel Gas* reckon that 25% of the carbon can be oxidized with water and present the following computation as to the efficiency of the primary combustion effected in that way: Three pounds of carbon burned with O of air and 1 lb. burned with O of water develop 17,600 B. T. U.; 1·5 lb. water, which furnish 1·33 lb. O to combine with 1 lb. of carbon, absorb by dissociation 10,333 British thermal units; the gas consisting of 9·333 lb. CO, 0·167 lb. H and 13·39 lb. N, heated to 600° F. absorbs 3748 heat units; 10,333+3748=14,081 and 17,600 − 14,081=3519 lost by radiation and otherwise, wherefore there is a

[1] Manufacturers of the well-known Taylor gas producer.

recovery of 10,333÷17,600=58·7% of the heat of primary combustion, even if all the sensible heat of the gas be lost. Under the above theoretical conditions 4 lb. of carbon would yield 338·9 cu. ft. of gas at 62° F. and atmospheric pressure, of which the composition by volume would be 37·31% CO, 9·35% H and 53·34% N. Since the heat absorbed in the dissociation of the steam is recovered in the subsequent combustion the gas of 4 lb. of carbon must carry $(4\times14{,}500)-(3748+3519)=50{,}733$ B. T. U. or approximately 87% of the original calorific energy of the carbon, without crediting the gas with its sensible heat or charging it with the heat required for the generation of steam, i.e., the carbon required to evaporate 1·5 lb. of water. As a conveyor of energy this gas has the advantage of carrying 4·46 lb. less of nitrogen than if the fourth pound of carbon were burned with air.

So high an efficiency cannot be attained in practice, partly because of loss of carbon which escapes combustion, partly because some of the carbon is burned to CO_2 instead of CO, and partly because some of the steam escapes decomposition. These elements of loss vary under different conditions. There must be deducted also the consumption of carbon for generating the steam. Assuming that the accessory plant of steam boilers evaporate 10 lb. of water per pound of carbon the production of 1·5 lb. of steam required for the oxidation of one pound of carbon in the producer necessitates the combustion of 0·15 lb. of carbon under the boilers, i.e., 0·15÷4=3·75% of the fuel is used in making steam and that loss is not recovered. The percentage varies of course with the efficiency of the boiler and the kind of coal.

The use of steam in the producer presents the further practical advantage that in cooling the zone of combustion the trouble from clinkering of the ash is reduced. In many cases this is a highly important consideration. The effect of blowing a producer with a very large volume of steam is shown by the results of the Mond producer, q. v. The gas from that is developed at a very low temperature and is consequently high in carbon dioxide, but because of its high tenor in hydrogen it possesses a great calorific power.

Practical Considerations in Fuel Gas Production.—The zones of combustion and distillation in a gas producer are illustrated in the engraving of the Wellman producer on page 323. The zones are not so distinct, however, in practice as is indicated by the illustration. The complete distillation of coal is attained only after long exposure to a high temperature, owing to its tendency to coke into large masses which are not easily broken. Consequently, lumps of coke frequently pass through the fire and drop from the grate while still retaining some volatile constituents, and in the zone

of combustion there is not only a mass of burning coke but also considerable coal that has been only partly distilled. Thus it happens that while a certain proportion of the gases of distillation escape from the top of the fire without being subjected to intense heat, the remainder is exposed to the high temperatures and chemical forces of the zone of combustion. A series of complicated processes of dissociation and synthesis is the result, in which the length of time and degree of heat are important factors.

Under otherwise equal conditions, a hot and deep fire will best promote the desired reaction, but even with a bed of incandescent coke 10 ft. in depth the escaping gases will still contain carbon dioxide. In practice the depth of the fire cannot be much over 6 ft. If deeper, it is found that no matter how thoroughly the upper surface may be stirred, the lower part of the fire is not thoroughly broken up by the poking, and the zone of combustion becomes honeycombed with large cavities affording passage for undecomposed air and steam. This condition is most marked along the walls, and the intense heat produced on the interior surface of these coking chimneys causes clinkers to fuse to the brickwork. Practice therefore demands that the thickness of the fire be limited, that steam be used to avoid extreme temperatures, and that the mass be kept thoroughly broken by frequent poking.[1]

Composition of Producer Gas.—The producer gas made from Pennsylvania bituminous coal blown with steam will contain usually from 34 to 42% of combustible matter. A gas containing the lower percentage may have approximately the following composition: CO_2, 5·2%; O, 0·4%; C_2H_4, 0·4%; CO, 22·8%; H, 8·5%; CH_4, 2·4%; N, 60·3%; total, 100%; all percentages being by volume. A richer gas is represented by the following analysis: CO_2, 4·04%; O, 0·08%; C_2H_4 and CH_4, 3·61%; CO, 22·98%; H, 11·41%; N, 57·88%. The latter analysis is of a gas made from Pittsburg run of mine coal assaying 1·19% water, 36·05% volatile combustible, 55·76% fixed carbon, 6·26% ash, and 0·74% sulphur. The second gas contained 38% combustible matter; the percentage of the latter in gas made from Pittsburg coal may sometimes be as high as 42%.

Besides the above mentioned constituents of producer gas, steam is always present in quantities varying with the moisture of the coal and the condition of the fire. It never appears in ordinary analyses, however, since it is condensed in taking the sample. Sulphur is also present, either as sulphurous acid or sulphureted hydrogen, or possibly uncombined.

Tar and Rich Hydrocarbons.—Certain coals produce also large quantities of tar, which are deposited in the gas flue. Part of this product, as well as

[1] H. H. Campbell, Trans. Am. Inst. Min. Eng., XXII, 373.

some rich hydrocarbons, are decomposed by the heat of the fire, forming large quantities of soot. When the fire is very hot the amount of soot is three times as great as when it is cool, and it may be as much as 5 cu. ft. for every ton of coal consumed. The formation of soot causes expensive and troublesome delays for cleaning the flues.

It is claimed by W. H. Blauvelt[1] and W. J. Taylor[2] that the unstable hydrocarbons, which do not appear in the chemical analysis of producer gas, because they are condensed in the receiving tube, contribute essentially to the calorific value of the gas. In order that those rich and powerful constitutents may be utilized, Taylor says that producer gas should not be cooled below 300° F. (149° C.). H. H. Campbell considers[3] that the influence of those condensible compounds has been overestimated. He states that in the consumption of 8000 tons of coal during a run of eight months the quantity of tar collected was only 44,000 lb., or 5·5 lb. per ton of coal, which is 2·5 kg. per 1000 kg. of coal. The calorific value of the tar was 9670 calories, whence the calorific energy in the condensed tar was calculated to be only 0·3% of the total calorific energy of the coal. He points out that even if all this tar could be kept in the gas it would not be a clear gain, for when rich hydrocarbons come in contact with the hot checker work of a regenerative furnace they are decomposed and carbon is deposited, which carbon if small in amount is burned on the next reversal of the valves, but its energy is lost through the chimney. The same criticism does not of course apply to a simple gas fired furnace. In the case above cited there was no record of the weight of the soot accumulated. The soot which probably does not exceed the tar in calorific value is even more troublesome in choking the valves and regenerative checker work, so that with regenerative furnaces nothing would be gained by preventing its deposition in the conduit.

Mr. Campbell did not in these criticisms, however, refer to the advantage to bituminous gas because of its luminosity, due to the unstable hydrocarbons, which Blauvelt emphasized. In commenting upon Mr. Campbell's paper, Mr. George W. Goetz also expressed himself of the opinion that Mr. Campbell had overlooked the importance of hydrocarbons in securing a strongly luminous flame in the furnace. Mr. Goetz stated that a 10-ton open hearth steel furnace could not be brought up to a steel melting temperature by a producer gas made from coke, although its tenor in carbonic acid was below 2%; but as soon as sufficient soft coal was charged upon the coke to make the flame luminous, the furnace heated up rapidly.[4]

[1] Trans. Am. Inst. Min. Eng. XXII, 381.
[2] Ibid. XVIII, 868 et seq.
[3] Ibid. XXII, 375.
[4] Ibid. XXII, 681.

THERMAL EQUATION OF THE PRODUCER.—In the gasification of fuel there is a loss of calorific power from several causes, each of which, as well as the sum of all of them, will vary with the kind of fuel and the manner of gasification. The measurements with respect to the gasification of a certain bituminous coal which were made by H. H. Campbell and reported in his elaborate papers in the *Trans. Am. Inst. Min. Eng.*,[1] are illustrative of the manner in which such losses occur. The total loss reckoned under the five captions was 2879 calories, or 35%.

The coal which was used contained 82% carbon and 4·7% hydrogen. It was gasified on an open grate with a steam jet in the blast. The loss of carbon in the ash was 6% of the weight of the coal, wherefore the quantity of carbon gasified was 76% of the weight of the coal. The gas produced had the following composition: CO, 22·8%; H, 8·5%; CH_4, 2·4%; C_2H_4, 0·4%; CO_2, 5·2%; O, 0·4%; N, 60·3%; total, 100% Each kilogram of coal developed 4·56 cu. m. of gas. The gas had a calorific value of 1170 calories per cu. m., wherefore 1000 calories were developed by 0·855 cu. m. of gas, which required for combustion 0·869 cu. m. of air, producing 0·267 cú. m. of CO_2, 0·133 cu. m. of H_2O and 1·202 cu. m. of nitrogen, making total products of combustion to the amount of 1·602 cu. m.

The total heat value of the coal was 8198 calories per kilogram, of which 2879 calories, or 35%, were lost in the producer, the remaining 5319 calories being delivered to the laboratory of the furnace as potential energy in the gas. The losses in the producer per kilogram of coal were itemized as follows:

(1) *Unconsumed Carbon.*—Potential heat lost by unconsumed carbon in the ash amounted to 483 calories or 5·9% of the total heat generated; this is due entirely to unconsumed carbon, the physical heat of the ashes being nearly all utilized while lying in the lower part of the fire and in the ash pit in heating the incoming air and moisture; the amount of heat actually carried away is specific heat and may be neglected.

(2) *Latent heat of volatilization of the hydrocarbons* amounted to 600 calories or 1·3% of the total heat generated.

(3) *Decomposition and heating of steam* absorbed 356 calories, or 4·3%; the loss of heat under this caption is caused in three ways, namely, (*a*) evaporation of moisture in the coal and heating of the resulting steam, which led to a loss of 18 calories, the coal containing 2% moisture, (*b*) heat absorbed by undecomposed steam, which amounted to 38 calories, and (*c*) decomposition of steam, which amounted to 300 calories, making the total of 356 calories.

[1] Vol. XIX, pp. 128 to 187, and Vol. XXII, pp. 345 to 511.

(4) *Radiation and conduction,* which are estimated by difference, caused a loss of 355 calories, or 4·3% of the whole.

(5) *Sensible heat of the gas,* assuming that the gas has a temperature of 750° C. and a specific heat of 0·317, amounted to 1085 calories, or 13·2%.

Gas Firing Compared with Direct Firing.—In comparing gas firing and direct firing, the loss of heat through unconsumed carbon will probably be considerably less in the gas producer than on the direct grate, especially if the producer be of the modern type with closed ash pit and no grate at all, the zone of combustion resting on the bed of ashes; in such producers, with certain kinds of fuel, the quantity of unconsumed carbon in the ash may be as low as 0·5% of the original carbon in the coal; on the other hand, with open grates, both in gas producers and direct heating fireplaces, the loss of carbon in the ash may be very high, especially with inferior coals.[1] The difficulties involved in the burning of bituminous coal by direct firing, which were discussed in a previous section, are overcome in gas firing, with the result that combustion is more complete.

Gas firing as compared with direct presents another important advantage in the fact that in general the quantity of air required for the combustion of the coal is considerably less, wherefore the heat is saved that would be otherwise consumed in raising the surplus to the temperature prevailing in the laboratory of the furnace. The necessity for less air for combustion in gas firing is due partly to the replacement by steam of a part of the air, possibly to the extent of one fourth of the oxygen required for the primary combustion to carbon monoxide in the producer; partly to the fact that the producer gas can be burned secondarily with a less excess of air than is required for the direct combustion of coal, and partly to the non-admission of false air during stoking. If the gas firing be done recuperatively, and the products of combustion are finally discharged to the chimney at a low temperature, the loss of heat through use of even a large excess of air is relatively small.

Position of Gas Producers with Respect to the Furnace.—A theoretical discussion of the subject of fuel gas production leads naturally to a consideration of the question as to the proper position of the gas producers with respect to the furnaces in which their gas is to be employed for heat-

[1] Geo. A. Davis reports, in Handbook of Chemical Engineering, II, 71, a sampling of ashes from a large steam plant in Great Britain which showed 27% C. In another case (op. cit., I, 235) a slack coal which contained 36·3% ash by analysis gave a refuse from the grates which contained 38% C., a fact which was only discovered through the complaint of the stokers that they had to "wheel out as much clinker as the coal they put in." Experiments in the laboratory of Coxe Bros. & Co. showed that in burning anthracite culm the carbon in the ash was sometimes as high as 58%.

ing. It is impossible to lay down any general rule covering this question, which must be answered for each particular case after a study of the conditions. The theory of fuel gas production implies that if the gas be cooled before combustion, there will be a large loss of thermal energy and the only possible advantage in gas firing must be in the utilization of such a high percentage of the energy delivered in the gas to the furnace that the ultimate result will be a higher development of the energy present originally in the fuel than can be obtained by direct firing.

It is evident that if the gas producer be directly connected with the heating furnace, as is the case with the furnaces used for the manufacture of illuminating gas, and the Boëtius and other furnaces of that type which are still employed extensively on the continent of Europe, there need be theoretically no loss of energy, inasmuch as the heat of the primary combustion is utilized directly in the furnace and there can be no condensation of tar or deposition of soot. The combustion of coal in such systems of gas firing in fact differs from the combustion on direct grates only in degree. On the other hand, although they utilize the heat of formation of carbon monoxide, they lose many of the advantages which are gained in independent producers at the expense of that heat. As compared with direct firing they gain only in ability to elongate the flame, together with possibly a reduction in the necessary supply of oxygen for combustion and consequent saving of heat units, and a greater facility for the adoption of a heat recuperative system. There is but little economy in labor and the temperature of the furnace is subject to fluctuation because of the intermittent character of the stoking, much as it is with direct firing. There is obviously no advantage in blowing such a producer with steam, apart from such physical advantage as may be obtained when badly clinkering coal is being used, because the heat of formation of carbon monoxide is being utilized directly in the furnace, and as has been pointed out previously, the introduction of steam cannot possibly lead to any gain in energy; in fact, the steam itself necessitates a certain consumption of coal for its production.

Grouping of Producers in Batteries.—When the producers are separated entirely from the heating furnace, it is possible and frequently advisable to group them in a battery, which makes it easy to deliver mechanically to them the coal which is to be consumed, and generally permits of economy in labor. It must not be expected, however, that gas firing will in any case lead to a great saving of labor, because the management of a producer involves a high class of stoking; it may be reckoned that the care of two 8 ft. circular producers, gasifying each about eight tons of coal per 24 hours is all that one man working a 12-hour shift is capable of. A great advantage

in the arrangement of the producers in battery lies in the fact that when installed so as to discharge into a common main with branches to the furnaces, the irregularities of gas production in the various generators are offset, with the result that the furnaces will receive an even character of gas.

If the heating furnaces are of the Siemens regenerative type there will be no gain of energy by supplying them with hot gas, and in connection with that type there is probably no question that the producers are best grouped in a battery at a convenient place within a reasonable limit of distance from the various heating furnaces and operated with the maximum quantity of steam so as to produce a gas with the highest possible content of hydrogen. If the heating furnaces be on the other hand of the counter-current recuperative type, or of a non-recuperative type, the position of the producer is governed largely by the question as to whether the gain in fuel energy is offset by the disadvantage of irregularity in the gas supply or vice versa. There are some heating furnaces, for example certain zinc distillation furnaces, which are so large that they require the gas from several producers, in which case it is possible to install the latter in small batteries close to the furnace, thus preserving much of the sensible heat of the gas and furnishing the latter in a fairly regular supply, without decomposition of the unstable hydrocarbons.

No Economy in Supplying Hot Gas to Siemens Furnaces.—It is now generally accepted among gas firing engineers that the producer should be placed near the furnace, if there be no offsetting advantages for a remote location. This principle applies, however, only to direct gas firing and does not hold good if the heating furnace be of the regenerative type. The difference between a regenerative and a non-regenerative furnace is frequently lost sight of in this respect. With a regenerative furnace if the producer be placed closely thereto, so that a hotter gas is delivered to the furnace, the only result will be a corresponding rise in the temperature of the products of combustion escaping into the chimney. It is true, says Campbell, "that when the gas is hot, less heating of the gas chambers is required, and hence less checker work will suffice; but that is a small matter, having no bearing on the fuel economy. Whatever is gained by hot gas at the incoming end is lost on reversal in the outgoing products of combustion. Moreover, a special system of valves must necessarily be constructed to handle hot gases, since ordinary valves soon warp and leak and water cooling is not to be thought of, for that involves chilling the gas, which is manifestly opposed to the intent of the practice in question."

Kinds of Fuel Suitable for Gas Production.—In the United States it is rare to find any fuel employed for the production of generator gas other

than coal, bituminous or anthracite. Gas firing in this country is practiced chiefly in the iron and steel districts of the East. In the metallurgical districts west of the Mississippi River there are only a few examples of the use of producer gas. In Europe, where gas firing is more generally practiced, fuels of various kinds are used, including wood, peat, lignite, brown coal and stone coal, and the design of the producers varies essentially according to the kind of fuel; thus the thickness of the bed that can be carried is governed chiefly by the physical condition of the fuel, while the form and size depend greatly on its purity.

Desirability of Good Fuel.—In some treatises on gas firing the reader is caused to understand that the fuel required for the production of gas in a generator need not be of superior quality. It is true that gas can be made from inferior fuels such as peat, wood and even sawdust, and these materials may be burned for the production of high temperatures in metallurgical furnaces, as is practiced at various places in Europe, where if they had to be burned directly the results would not be satisfactory. In such work the gas producer plays the part of an apparatus for recovering the energy of what would be otherwise a waste or useless product. The producer cannot, however, create any energy beyond what exists originally in the fuel, and it follows that a better gas can be made from a high grade fuel than from a low grade. A true statement of the case is this: although the producer may make it possible to use an inferior fuel, which perhaps cannot be burned satisfactorily by direct firing, better results are obtained with a fuel of superior quality, precisely as in direct firing. There are, however, some kinds of fuel of fairly high calorific power which possess certain qualities, for example, a badly clinkering ash, which gives so much trouble in the producer that they cannot be satisfactorily gasified therein, although they may still be useful on a direct heating grate.

In places where two kinds of fuel are available for gas production the choice is governed by the economical considerations, including the first cost of the fuel, the quantity and calorific power of the gas developed, and the cost of managing the producers; for example, it is the experience in the iron and steel district surrounding Pittsburg, Pa., that run of mine coal is preferable to slack for gas production, notwithstanding the higher first cost per ton of the former. In this connection it may be remarked that the difference in the price of run of mine coal and slack is frequently less in proportion than their actual heating powers. Aside from that point, the run of mine coal is apt to permit a better management of the producer, being not only lower in ash, but also capable of a more gradual development of gas, because of the slower distillation and combustion of the large

lumps of coal which exist in it, whereas those actions take place more rapidly with fine coal slack and there are greater rises and falls in the evolution of gas in the intervals between charging.

Difference in Results with Good and Bad Coals.—In the pamphlet concerning the Taylor gas producer and gas fuel, published by R. D. Wood & Co., of Philadelphia, the character of fuel best suited for gas production is summarized as follows:

"In making gas from bituminous coal the best results are obtained from a good clean coal, low in ash and high in volatile matter. A poorer quality does not make so good a gas, nor can the producer be driven so hard. In high temperature work a high percentage of volatile hydrocarbons in the coal is very desirable and a smaller consumption of coal is then needed to do a given work. Thus where coals are inferior and cheap, it may be cheaper to bring from a distance higher priced coals of good quality. This is done advantageously in numerous instances.

"The size of coal is not so important, especially when the coal cakes, for it then fuses together in large masses, which on being broken with a bar make the fuel bed porous and open. Nut size is a very convenient one for use in the producer, although run of mine in which the lumps are small enough to pass through the hopper, or slack, or a mixture of the two, are used very successfully. The clinkers which form from soft coal are rarely large and are handled with little trouble, except when very lean coal is used.

"Although the producer works to best advantage on coal of good quality, nevertheless the modern facilities for cleaning the producer and the perfect application of the steam and air make it possible to use successfully a very inferior coal, but the gasification must be slower than with good coal. There is one large plant using Taylor producers to gasify a slack coal containing over 40% of ash, and what is worse, a large percentage of iron sulphide.

"When anthracite is used the cheapest form is No. 1 buckwheat, with a low percentage of difficultly fusible ash, low in moisture and high in volatile combustible. An important point in using anthracite is that too much fine dust is very objectionable, since it makes the interstices too small, or much smaller in some parts of the bed than others. This tends to honeycomb the fuel bed unless much barring is done, or what is worse, if the resistance in the fuel bed is too great, the blast will seek the walls as the place of least resistance, and the gas will be worthless, becoming high in carbonic acid. Anthracite in the form of culm or poorly prepared buckwheat cannot be gasified to advantage in a producer. However, a continu-

ous feed adjusted so as to maintain a proper fire surface, showering the coal evenly and regularly into the producer, assists largely in the use of those coals which are inferior because of size or quality."

COMPARISON OF ANTHRACITE AND BITUMINOUS GASES.—In the previous discussion of the production of fuel gas reference has been made especially to gas produced from bituminous coal, which is industrially by far the more important kind of fuel gas. In the eastern part of the United States, however, where anthracite is available, gas is made very successfully from that kind of coal, and is used not only for heating purposes but also for the operation of gas engines, for which purpose bituminous gas made in the ordinary manner is not to be recommended, because of the difficulties that are experienced on account of its tarry constituents. Because of the absence of those constituents in anthracite fuel gas, the latter presents important differences from bituminous gas even when used for heating purposes. This subject has been discussed lucidly by W. H. Blauvelt in the *Trans. Am. Inst. Min. Eng.*, XX, 625, and XXII, 380. The following is summarized from Mr. Blauvelt's papers:

With producers operated by natural draught it is necessary to use the larger and more expensive sizes of anthracite in order to permit the draught to pass through the fuel bed with sufficient rapidity. Attempts to use "buckwheat" and "rice" with forced draught have met with but partial success in most types of producers, one reason being that when the small coal is incandescent it will run through the smallest opening in a persistent stream, occasioning a large loss through the grate when the producer is cleaned and also injuring the quality of the gas for a considerable time. That difficulty has been overcome, however, in the later forms of producers which have no grates and with them it is possible to make from anthracite down to buckwheat size a good gas, low in CO_2 and with but trifling loss of carbon in the ash. With smaller sizes, however, every attempt to force the rate of combustion to a reasonable speed results in deflecting the steam and air blast to the walls, thus creating an annular zone of intense combustion, with an unburned center, and producing only a small quantity of gas containing a high proportion of CO_2.

Failure of Ordinary Analysis to Show the True Composition of Bituminous Gas.—A notable difference between anthracite and bituminous gas is their appearance when burning. The former burns with a bluish, almost colorless flame, whereas the latter burns with a luminous flame. The composition of the two gases as shown by chemical analysis does not indicate the reason for that difference in appearance or the difference in heating effect which is experienced. Representative gases made from anthracite and

bituminous coal, according to the best practice, show substantially the same composition as appears in the following analyses:

	CO	H	CH_4 C_2H_4	CO_2	N
Anthracite....	27·0%	12·0%	1·2%	2·5%	57·3%
Bituminous...	27·0	12·0	2·5	2·5	56·0

It is understood now, however, that an analysis of bituminous gas does not represent its actual composition. Theoretically a gas made from bituminous coal containing 30 to 35% volatile matter should contain 12 to 14% of hydrocarbons, or six times the quantity shown in the above analyses. The reason that the gas does not show such a tenor lies in the fact that when bituminous coal is gasified, only a small portion of the hydrocarbons is converted into fixed gas, the remainder being merely volatilized and carried as vapor to the furnace. That vapor is condensed in the apparatus for sampling and therefore does not appear in the analysis. This circumstance indicates the importance of maintaining the temperature of bituminous gas during its passage from the producer to the furnace at a point which will prevent the condensation of the hydrocarbon vapors, in order that the full value of those may be secured.

Superiority of Bituminous Gas for High Temperature Work.—According to W. J. Taylor[1] a bituminous coal containing 32% of hydrocarbons, 55% of fixed carbon and 13% of water and ash, should yield theoretically a gas containing by volume 27·8% CO, 2·7% CO_2, 13·2% CH_4 and C_2H_4, 8·3% H and 47·8% N. Such a gas has a calorific power of 58 calories (230 B. T. U.) per cubic foot, while the same volume of the best quality of gas from anthracite (for example, one of the composition mentioned above) has only about 34 calories, a difference of over 60% in favor of the former, although the coals from which the two gases were made show a thermal difference of only 7%, containing respectively 7900 and 7400 calories per kilogram. These figures explain why bituminous gas is so much superior for heating at high temperatures while in low temperature work its superiority does not appear; in fact, at low temperatures the values of the two gases are nearly equal when the gas from a given quantity of coal is taken as a basis of comparison.

In boiler firing, where only a low temperature is reached, the work done by a pound of bituminous coal is but slightly greater than that done by the same quantity of anthracite. The reason for this is because the shell of a boiler cannot be heated beyond a point which is far below the flame tempera-

[1] Trans. Am. Inst. Min. Eng., XVIII, 870.

ture of the gas, and the work done is simply the transference of the heat units from the gas to the water in the boiler. It is, therefore, no advantage to bituminous gas that the heat units are more concentrated than in anthracite gas. In estimating the work of the fuel the only question is: How many heat units does it contain that are capable of absorption by the water, assuming the boiler to be set so as to absorb all available heat from the gases. In a high temperature furnace the material to be heated is usually charged cold and the object is to raise it as rapidly as possible to a high temperature. Under those circumstances the difference in intensity of 60% in favor of the bituminous gas is a great advantage. A comparatively small portion of the heat generated is absorbed by the charge and the gases leave the furnace without being materially cooled. It is apparent, therefore, that intensity is desirable rather than the mere delivery of a certain number of heat units to the furnace per minute, and that the inferior intensity of anthracite gas can be offset by its consumption in greater quantity only to a point where the necessarily increased size of the furnace causes the heat to be radiated as fast as it is supplied.

Difference in Flame Temperature and Luminosity.—Theoretically the maximum temperature of the furnace should be that of the flame temperature, providing that all radiation could be prevented. Assuming the gas and air for combustion to enter at a temperature of 1000° C., the flame temperature of anthracite gas would be approximately 2850° C. and that of bituminous gas 3190° C. Although those temperatures are much higher than can be attained in practice, they are assumed to show approximately the difference obtainable with the two gases, since with properly constructed furnaces the loss by radiation would not differ materially with such difference in temperature. The difference in flame temperature is not sufficient, however, to account for the great superiority of bituminous gas, and the superiority of the latter must be attributed largely to the fact that the gas being more concentrated burns more quickly, thus developing a higher heat before it can be dissipated by radiation, and also to what is probably more important, the luminous quality of the gas, which causes heat to be given out to the charge by radiation as well as by conduction. The importance of heating by radiation is now acknowledged and the accepted type of high temperature furnace is one with a high roof and ample room for the development of the flame to the highest temperature. The flame is relied upon to heat the charge by its radiant power, no attempt being made to force it down upon the charge as with the old low-roof furnace. The superiority of bituminous gas over anthracite is therefore due largely to its hydrocarbons which give it intensity and also luminosity.

HEATING BY RADIATION.—The questions of the luminosity of burning gas and heating by radiation were discussed very lucidly by George W. Goetz in the *Trans. Am. Inst. Min. Eng.*, XXII, 682 to 685, in connection with which some important views imparted in conversation by Friedrich Siemens, of Dresden, were put on record. Before stating Mr. Siemens' views, Mr. Goetz referred to several considerations, mentioned in a debate on this subject before a number of British gas engineers, as follows:

"It is an undoubted fact that the heat of the Bunsen burner is more highly concentrated than that of the illuminating flame, and it is, therefore, universally adopted for boiling purposes where it is desirable to concentrate the whole heat on the vessel. For this purpose the Bunsen flame possesses another very desirable qualification, viz., it does not deposit soot on the bottom of the vessel against which it is directed, and thereby does not obstruct the conduction of the heat by the intervention of a powerful non-conductor. But beyond that, the superior qualities of the Bunsen flame do not go."

Mr. Goetz went on to say that if the statement that the Bunsen burner develops a much higher temperature than an illuminating flame were meant to be accepted as an absolute one it could be proved by an experiment suggested by Doctor O'Conor Sloane that such is not the case, Doctor Sloane's statement being as follows:[1]

"Take two burners (a Bunsen and a white flame) adjusted to consume equal quantities of gas, and introduce them into two separate tubes surrounded by a water jacket; the same amount of water at the same temperature to be placed in each.

"Let each tube have sufficient convolutions to present a large conducting surface. Allow the flames to burn the same length of time in each tube and take the temperature of the water. It will be the same in each case, proving the heat developed in both burners to have been equal."

If these experiments were tried under an open vessel, with Bunsen and white flame respectively, the results would be very different. The Bunsen flame would boil the water more quickly than the white flame, because a large part of the heat developed by the latter would never strike the bottom of the pan at all, but would be radiated off in all directions. Quoting Doctor Sloane's words, "the difference between an luminous and a non-luminous flame is one of character only; if all the heat be utilized, one is as efficient as the other." Professor Tyndall says:[2] "The Bunsen flame is much hotter than an ordinary flame, because the combustion is much

[1] Journal of Gaslight, XXXIII, 878.
[2] Heat as a Mode of Motion, fifth edition, Longman, 1875, p. 49.

quicker, and therefore more intense, but it is not hotter, not nearly so hot, to a body exposed to its radiation." Tyndall gives the proportions of radiant heat as 30% in the illuminating flame and 12% in the Bunsen flame.

Opinions of Friedrich Siemens.—Mr. Goetz then presented the statements of Friedrich Siemens concerning the general application of heat as follows:

"In all heating operations it is the natural tendency to try to get the highest effect with the least consumption of fuel and labor. To attain the above, perfect combustion is essential; yet perfect combustion does not cover all practical requirements. There are heating operations which require particular conditions. One must consider particularly whether a large quantity of heat is required, as when firing a boiler, or whether intensity is required; or whether both intensity and quantity are wanted, as in a steel melting furnace.

"One should not attempt to burn gas according to a fixed scheme; each case ought to be specially examined and the design so made as to meet all practical requirements. To insure perfect combustion every flame should have ample room for free development.

"Practical observations have shown that a flame must be so directed in a furnace that it will not come in contact with the surface of a solid, such as the walls of the furnace, or the material which is being heated therein.

"When ample space is given to a flame for its free development, not only a higher effect is attained, but also a saving in refractory materials, time and labor. Recent constructions have shown the practical value of these facts, especially in the designs of steel and glass melting furnaces with high roofs.

"One of the advantages of gaseous over solid fuel is that the former permits a more accurate regulation of the air for combustion.

"The Bunsen burner is only of advantage when small surfaces are to be heated or when the flame is to strike a body, as shown by the incandescent gas light. The Bunsen burner is especially adapted to certain small operations in a laboratory, in the household, or in the kitchen, but not to heat a large space and extended surface, as in a large furnace. In such cases a large luminous body of flame is essential for the production of radiant heat.

"The flame of the Bunsen burner is non-luminous, because there is no free carbon in it; consequently it radiates but little heat and can therefore heat by direct contact only. Although the Bunsen burner seems so simple and advantageous from a theoretical standpoint (as it is, without doubt, for certain cases), it is useless for most technical purposes, especially metallurgical operations on a large scale.

"When gas is consumed in a large furnace the gas and air must be brought

together in such a manner that a flame with a high radiating power is produced; for thus the heat is best transmitted to the sides and walls of the furnace and to the material to be heated.

"Furnaces fired with solid fuel produce a more or less radiating flame, according to the nature of the fuel. Practice has shown that only such coal is good for direct firing in a reverberatory furnace as gives a large amount of hydrocarbon gases, for these gases increase the radiating power of the flame. The very best coke is useless for the same purpose, although the highest temperature can be produced by it when the body to be heated is placed in direct contact with it, as in crucible steel practice.

"There are no special difficulties in the production of radiant heat in the combustion chamber of a direct-fired furnace, provided the proper fuel be selected; but the case is not quite so simple when gas is to be used in a regenerative furnace. It is of the greatest importance how the gas and air are brought together in a regenerative gas furnace, according to the quality of the gas and the use of the furnace. When the mixture takes place too quickly and completely a very short flame is produced, which is intensely hot, but only slightly luminous and consequently possesses very little radiating capacity. When the reverse is the case—that is, when the mixture is very incomplete—good combustion cannot take place; and, besides a loss of heat, a number of other evils will arise, which will be mentioned below. For good practice, and to secure at the same time a flame with high intensity and a high capacity to radiate heat, a condition between these two extremes must be maintained.

"How gas and air must be brought together to produce radiant heat depends upon the quality of the gas, the desired temperature, the size of the combustion chamber of the furnace, and the special operation to be performed. In general it is best to admit the gas through a horizontal slit in the end of the furnace, over which the air is admitted through a similar slit, although somewhat larger in section. As the gas has a considerably lower specific gravity than the air, it has a tendency to rise through the air, whereas the air sinks through the gas and thus a sufficiently rapid mixture of the two takes place. Good results from this arrangement depend upon the temperature of the two gases and the relation of their temperature to each other, as the specific gravity is changed by heat.

"The light-and-heat radiating capacity of a flame depends upon the presence of free carbon particles, produced therein by a dissociation of hydrocarbons. This free carbon, although infinitely minute, is present as solid particles in the flame, and is heated up to the flame temperature. Every carbon particle radiates heat and light to all sides, and continues to do so

until it is burned to carbonic acid, or until the flame is cooled to below the temperature of incandescence. The free carbon in the flame is always the last which is consumed, so that in all cases of incomplete combustion it is this carbon which remains and deposits itself as soot in the flues, or passes out of the chimney, with the products of combustion, as black smoke. These small bodies, or particles, which give to a flame the capacity to radiate heat, and which give such a high efficiency when properly utilized, cause the disagreeable smoke in our cities when the furnaces are not properly constructed or managed."

Special Conditions of the Distillation Furnace.—The zinc distillation furnace differs from the open hearth furnaces inasmuch as its combustion chamber is to a large extent filled by retorts, which can be heated properly only when exposed directly to the burning gas; otherwise radiant heat is obstructed by intervening retorts. It is essential, therefore, that all the retorts be enveloped by the flames, and there be room for the latter to develop properly. In the Silesian and Rhenish furnaces the arrangement of the openings in the hearth to the flues insures that the flames will be divided between the various ranks of retorts. In the direct-fired Belgian furnaces the position of the grates directly under the retorts causes the whole combustion chamber to be filled with burning gas and no special means of circulation are required, the two simple openings in the arch of each single furnace being usually sufficient. In long gas-fired furnaces in which the movement of the burning gas is longitudinal a mixing chamber is frequently provided at the inlet end, which affords room for combustion to be well established before the gas is divided between the rows of retorts. In some furnaces low bridge walls rising from the hearth at various places are found to be advantageous.

TYPES OF PRODUCERS.—The number of different gas producers that have been designed and used since Bischof built his first one at Mägdesprung in the Harz, in 1839, is very large. There have been many eccentric forms and methods of operation, few of which have survived, but aside from those the standard producers from the earliest time may be considered broadly as falling in one of two classes; namely (1) the step grate, natural draught generator, exemplified in the Siemens and Boëtius types, which is nothing but a development of the ordinary fire box; and (2) the shaft furnace, with or without a grate and worked by natural draught or by forced draught. Producers of the latter type, when without a grate and blown from the bottom are practically identical in many respects with blast smelting furnaces; the blast smelting furnace is in fact a gas producer, and in the metallurgy of iron is utilized as such, besides for the performance of its chief function.

The analogy between this type of gas producer and the smelting furnace is exemplified most strikingly in certain producers wherein a flux has been mixed with the coal to form with the ash a slag which was permitted to run continuously from the bottom.

In modern gas firing the shaft furnace type of producer is most commonly used, although the step grate type still finds extensive application, especially in direct connection with the furnace to be heated. In both types the modern improvements have been rather in details of construction and a proper adjustment of dimensions than in any radical change of principle. Many of the supposed innovations are really nothing but revivals of ideas that were tried early in the history of gas firing. In the following brief historical review it will be observed that the boshed sides of the shaft furnace, the discarding of the grate, the water seal, the mechanical removal of the ashes and the use of steam are all of comparatively ancient origin.

Bischof's first generator (1839) had the form of a small blast furnace, but it had a grate and was worked by natural draught. In 1840 Ebelmen built, at the Audincourt iron works in France, a generator which had the form of a blast furnace, no grate, and was blown with air; at first there was trouble from clinkering of the ash, but by the introduction of a suitable flux with the coal the ash was converted into a fusible slag, which ran out from the bottom of the producer. In 1868 E. Minary proposed again to dispose of the ash in that manner. In practice, however, it has not been found advantageous either to the operation or to the life of a producer to raise the temperature of primary combustion sufficiently high to melt the ash, apart from the inconvenience of mixing fluxes with the fuel.

The use of steam for blowing gas producers is generally credited to Wilson, who patented in 1876 a type in which the blast was supplied by a steam jet entering the narrow throat of a tapered vertical pipe, inducing a current of air in proportion to the quantity and velocity of the steam. This producer had a closed hearth and no grate bars. The blast was delivered in the center, so that contact of both air and steam with the fuel was ensured. Originally the shaft was made rectangular, but that form was soon abandoned in favor of the circular, which was built inside of a cylindrical iron shell. The gas issued from the producer at a pressure equivalent to 1 to 3 in. of water and the coal was burned at the rate of 25·7 lb. per sq. ft. of hearth per hour as compared with 12·5 lb. which is the usual rate with the Siemens producer. In later patents Wilson described a form of producer for continuous mechanical discharge of the ash, the bottom of the shaft being closed by means of a water seal.[1] The Wilson producer was

[1] British patent, No. 4697 of 1882; Journ. Soc. Chem. Ind., II. 453.

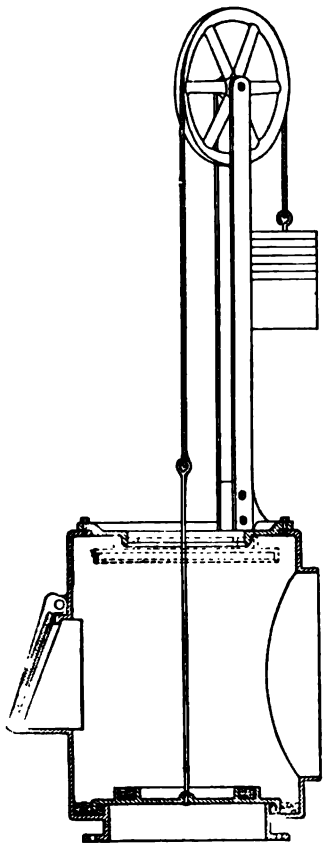
probably the forerunner of the modern type, but the use of steam in connection with it was not original. Sir William Siemens early proposed to blow his producer with steam, and in 1863 Siemens producers were constructed with closed ash pits and injection of air by means of steam jets. Wilson did not originate the idea, but he used a higher steam pressure than did the Messrs. Siemens.

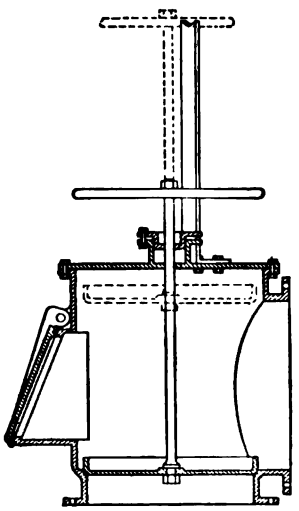
Cylindrical iron casings for the shaft were used by G. H. Benson at Cleckheaton, England, in 1868. His producers were blown with hot air, while steam superheated by a coil of pipe in the brick lining was introduced at various points in the circumference of the bosh near the bottom. The bosh itself was made of cast iron water jackets to avoid difficulty from clinkers.

Design of Producers.—The modern shaft producer has a water-sealed bottom and no grate. Such closed hearth producers can be operated without stoppage and are as amenable to the action of the poking bar as are producers with grates, while they are free from the disadvantage of the latter in being unable to use fine coal and of losing a considerable quantity by dropping between the bars during stoking and cleaning; moreover, the cost of renewals is reduced by the absence of grate bars, and it is easier to maintain an even bed of fuel than on a grate unless the latter be a simple horizontal one. Combustion is carried on at the maximum economical rate by means of forced draught, which has all the advantages over chimney draught that it does in ordinary direct firing. Steam and air together are commonly used and the blast is almost always introduced at the center of the shaft. It has been found repeatedly that when the blast is delivered at the sides or walls of a producer a portion of the air and steam will pass up their surfaces without combining with the fuel and the resulting gas contains uncombined oxygen besides a larger proportion of nitrogen than is necessary. To avoid that difficulty as well as to concentrate the combustion in the lower part of the shaft the walls are frequently boshed in.

A gas producer of rectangular shape, without an enclosing shell, should have walls of fire brick at least 13·5 in. thick and must necessarily be securely supported by buckstaves and tie-rods. Save for the Boëtius generators gas producers are rarely built in that way nowadays, the common form being the cylindrical shaft, which is provided in the form of a shell of tank steel lined with fire brick. The bottom of the shaft dips into a cement-lined basin of water and is supported by four cast iron columns, standing upon a solid foundation. The shell is made of ¼ in. steel and the fire brick lining is 9 in. thick. The top of the shaft is covered in with a dome of fire brick 9 in. thick, above which there is laid a cast iron plate provided

with the proper openings for feeding in the coal and for the insertion of poker bars. These openings are generally water-sealed so that there will be no leakage of gas from the top of the producer. In putting in the fire brick lining and roof the same precautions must be observed as in general furnace construction. The fire brick should be of good quality and should be laid carefully with good joints, each brick being first moistened and then knocked





FIGS. 200 AND 201.
VALVES FOR GAS PRODUCERS.
Fig. 200: Sand Valve.
Fig. 201: Metal Seat Valve.

into place with light blows of a hammer. Lime mortar should not be used any where or under any circumstances; the brick should be laid in a paste of one part of milled fire clay tempered with one part of finely ground chamotte or with two parts of clean sharp sand, the constituents being intimately mixed and diluted with water until the paste has a syrupy consistency (vide the section on furnace construction in the next chapter).

The cylindrical form of producer is the easiest of all to keep in shape. The continuous enclosing envelop of rolled steel cannot be improved upon, and this construction may be considered as representing the maximum of strength. They can be made absolutely gas tight, and being built independently any particular producer can be repaired at any time. In a well constructed producer, however, there is so little to get out of order that the outlay for repairs and renewals ought never to be heavy; probably not to exceed an average of $50 per annum. Every generator should be provided with a tightly closing valve or damper, which will permit a careful regulation of the gas stream and should be placed as near as possible to the generator. Types of valves are shown in the accompanying engravings, from which their mechanism will be understood without further description.

The coal is fed into the producer by means of a bell and hopper, placed in the center of the top, which is so arranged that there will be the least possible escape of gas in the operation. In modern installations a large hopper full of coal is supported over each producer whence the coal can be drawn directly into the small hopper of the producer itself. The large hoppers are filled by mechanical conveyors. Such an installation is very conveniently and economically arranged when the producers are grouped in a battery; if set separately the supply of coal will probably be delivered best by means of a cable railway. When the producers are grouped in a battery they may be set as close together as desired, providing access be given to the ash pit from two opposite sides, which there will always be if the producers are set in a row. The large hoppers over the producers must be sufficiently high above the latter, or else be to one side instead of directly overhead, so that the long poker bars can be without difficulty inserted through the poke holes in the top of the producer. The poke holes must be arranged in such a way that all parts of the fuel bed can be reached conveniently.

Dimensions and Capacity.—The size of a producer is governed to a great extent by the work that is required of it, but in extensive installations it is generally desirable to build them as large as is consistent with perfect combustion of the coal and regular development of the gas, inasmuch as the larger a producer is the less is its relative first cost and the space occupied by it, the lower is the cost of labor in attendance per ton of coal burned and the lower the percentage of heat lost by radiation. Practically it has been found, however, that a circular producer 8 ft. in diameter inside, having a shaft area of 50·265 sq. ft. is about as large as it is advisable to build and the standard producers of most of the modern designs have that diameter, although in some instances it is exceeded. In height from the ground to the top, measuring outside, they are generally from 12 to 15 ft.

The capacity of a gas producer depends chiefly upon its grate, or hearth, area and the rapidity of driving, the latter factor being a good deal dependant upon the kind of coal. With certain kinds of coal, producers have been forced to burn as much as 25 lb. per sq. ft. per hour, which would be 15 tons per 24 hours for an 8 ft. producer. With other kinds of coal the rate may be as low as 12·5 lb., or 7·5 tons per 24 hours. In determining the rapidity of driving the quality of the gas and other factors have to be considered and so high a rate as 25 lb. per sq. ft. per hour is seldom aimed at in modern practice. With Pennsylvania bituminous run of mine coal it is generally considered that about eight tons per 24 hours is all that an 8 ft. producer should be required to burn and yield a good gas and perfect combustion of the coal. That is 13⅓ lb. per sq. ft. per hour. R. D. Wood & Co., in their pamphlet previously quoted, state that in ordinary service a producer of 50 sq. ft. area of fuel bed will on the average gasify eight tons of Pennsylvania or West Virginia coal or 6·5 tons of anthracite pea coal; these figures are respectively 13⅓ and 10⅝ lb. per sq. ft. per hour. The fusibility of the ash in the coal has a good deal to do with determining the maximum rate of combustion in a producer. With the most infusible ash about 15 to 16 lb. of coal per hour per square foot of fuel bed is probably the most that can be burned continuously so as to afford a good gas, though some of the Western lignites, which can be gasified at a much higher rate, are an exception to this statement; with a very fusible ash the rate of combustion must be reduced in order to make good gas continuously without excessive labor or much waste.

The cost of a modern 8 ft. gas producer is $750@$1000, erected, varying according to the special design and the market price for the material. This does not include the steam and gas connections, the operating platform or the shed, but provides the producer erected on its foundation, without those accessories. In estimating the cost of a gas producer plant it is important not to overlook the steam boilers for blowing the producers.

Management of Gas Producers.—The coal should be put in gradually and as regularly as possible; otherwise there will be irregularity in the delivery of gas. If a large quantity of coal be charged into the producer at one time there is likely to be a sudden evolution of gas, and since the dampers controlling the admission of the air for the secondary combustion in the furnace cannot be regulated every time the producer is charged, there may be at such times an excess of gas in the furnace, which will go up the chimney unburned. Between the chargings the quantity of gas generated decreases and there may then be an unnecessary excess of air in the furnace, carrying off heat. The bed of coal in the producer must be kept even, else

the air seeking the channel of least resistance will pass through the thinner parts, generating a poorer gas, which often carries an excess of carbon dioxide and continually varies in its composition. The bell and hopper feed, which is generally used, to a certain extent distributes the coal over the bed in the producer, but its action is intermittent and depends entirely upon the attendant. Uneven distribution can, of course, be remedied by poking, but that involves hard labor which is often shirked. The coal in the producer must also be poked to break up the cakes and lumps and thus preserve a uniform flow of gas.

C. W. Bildt has patented a mechanical feed for gas producers, which has been used successfully in practice. It consists of a coal holder resting upon and attached to the top of the producer, below which, i.e., inside of the producer, is placed a rotating distributing disc provided with specially constructed blades. The disc is rotated by a vertical shaft supported by the coal holder and driven by a worm gear. The disc makes ¼ r. p. m. The distributing blades are arranged spirally, so that as the disc is rotated coal is dropped evenly over all parts of the bed in the producer. Bildt states that the use of these feeders in various places has led to savings of 15 to 30% in coal and better performance of the furnaces because of the more uniform gas, while labor has been reduced because of the less poking required. The device is durable and requires few repairs.[1]

Boëtius Producer.—This well known form of gas generator is still largely used on the continent of Europe, commonly in direct connection with the furnace to be heated. As a system of semi-gas firing it is one of the easiest to arrange and is perhaps even now one of the best. The generator itself has a combination of step and plane grate and in its cross-sections resembles the standard type of Siemens producer. The specially distinguishing feature is the air passage surrounding the lining of the fire box. These passages connect with the external air, which circulates through them, cooling the lining and itself becoming heated thereby. The supply of air thus preheated joins the gas through ports in the short passage by which the fire box communicates with the laboratory of the furnace.

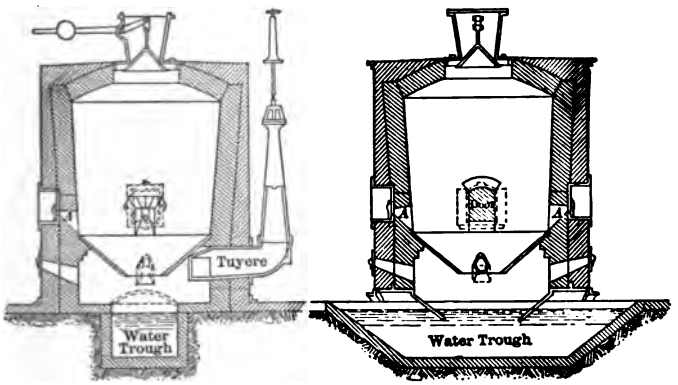
The advantage of this arrangement, as to which the success of the Boëtius generator in practice leaves no doubt, will be readily perceived from what has been said in a previous section with respect to the direct combustion of bituminous coal. There is no recuperation of heat because that which is acquired by the air is robbed directly from the fire and no prevention of loss except what is cut off from radiation into the outer brick work, but

[1]Trans. Am. Inst. Min. Eng. XXVIII, 166 to 176; United States patents, 442,676, Dec. 16, 1890, and 498,229, May 30, 1893.

heat is taken from a place where there is a surplus and transferred to another where it is advantageous, enabling combustion to be effected more completely than in ordinary direct firing.

The Boëtius producer has been used extensively by European zinc smelters. Its disadvantages are said to be the leakage of gas from the fire box into the surrounding air flues, through cracks in the walls, and the necessity for a good grade of coal. It is illustrated in Figs. 16 to 18, on pp. 23 and 24, and is shown in connection with various distillation furnaces in Chap. X.

Dawson Producer.—This producer which is used extensively in Great Britain is similar to the modern American producers, consisting of a circular brick shaft in a cylindrical steel shell, with a water-sealed bottom. Toward the bottom of the shaft there is a cast iron bosh, beneath which the



FIGS. 202 AND 203. DAWSON PRODUCER.
Transverse vertical sections on planes at right angles to each other.

air is blown in. The top of the shaft is roofed over in dome-shape and is provided with the usual bell and hopper feed. In one side of the shaft there is a door, or man-hole, which is used only when repairs are to be made.

Dowson Producer.—This is a simple cylindrical shaft of fire bricks, enclosed by an iron shell. It has a horizontal grate, a closed ash pit, and a blast of steam and air under the grate. The bottom is closed, but is not water-sealed.

Duff Producer.—The Duff producer is a square shaft of fire brick built up within a cylindrical steel shell; the angles are filled in so as to avoid sharp corners. The bottom of the shaft dips into a basin of water, forming a water seal. Above the water level there is a rectangular grate, sloping in ridge form. The air blast is introduced through a pipe entering the shaft

just under the grate. The top is arched over and is pierced with openings for the usual bell and hopper feed and a number of poke holes arranged

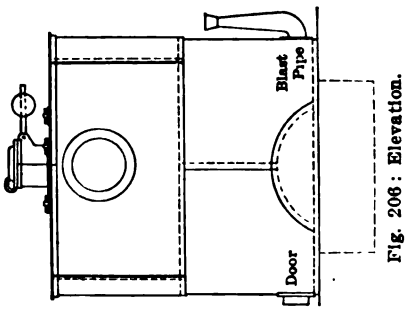


Fig. 206: Elevation.

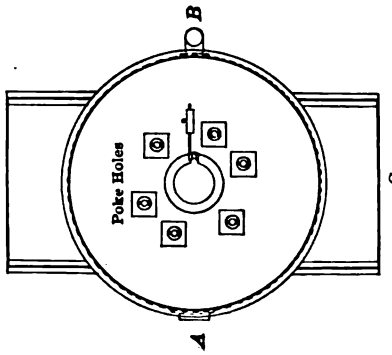


Fig. 209: Plan.

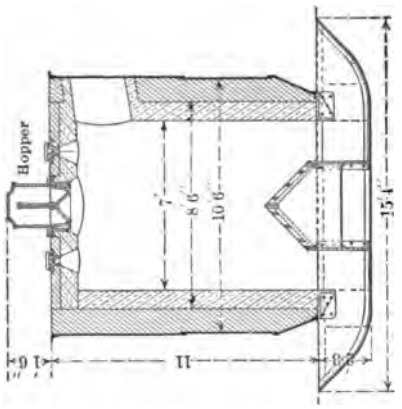


Fig. 205: Section CD.

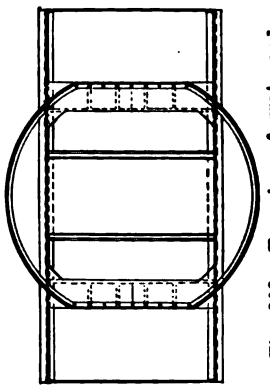


Fig. 208: Top view of grate and brick bearers.

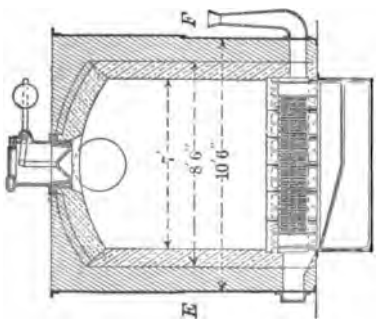


Fig. 204: Section AB.

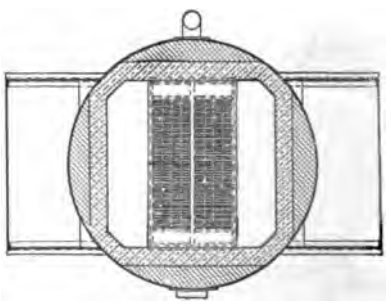


Fig. 207: Section EF.

FIGS. 204 TO 209. DUFF PRODUCER.

around the latter. The design and construction of this producer are shown clearly in the accompanying engravings. The purpose of the inclined grate

is to distribute the blast evenly under the bed of fuel and also to form guiding surfaces down which the ashes will slide toward the exterior.

The Duff producer is commonly built with a shaft 7×7 ft. in horizontal section; or 9×9 ft. It is claimed that the former will gasify 12 tons of Pennsylvania mine run bituminous coal per 24 hours, or 20 lb. per sq. ft. per hour; and the latter 15 tons, or 15 lb. per sq. ft. per hour. A large number of these producers are in use both in Great Britain and the United States. The American rights are controlled by the Duff Patents Co., of Pittsburg, Pa.

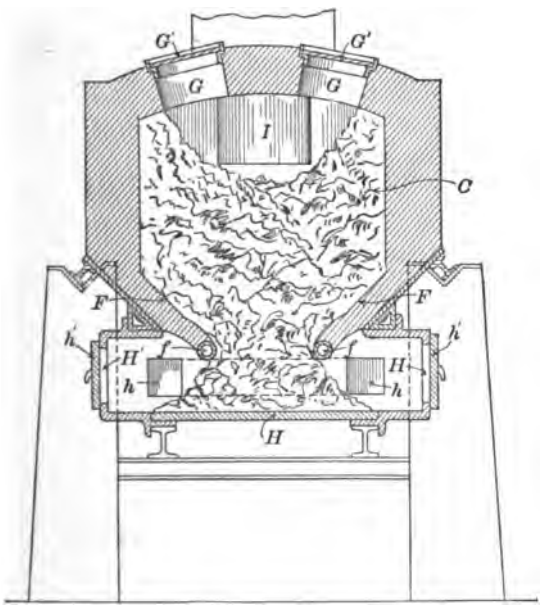


FIG. 210. HEGELER PRODUCER.
Vertical Section.

Hegeler Producer.—The Hegeler producer is in use at the works of the Matthiessen & Hegeler Zinc Co., at Lasalle, Ill. It is a brick shaft, square in horizontal section, contracted toward the bottom as shown in the accompanying engraving. The ashes, supporting the column of fuel, rest on a hearth *H* which is supported by two I beams several feet above the ground line. These I beams rest on two cross-girders, supported by four brick columns, which also support the shaft of the producer. The ash box is enlarged so that air from the blast flues *hh* can circulate around the pile of ashes resting on the hearth and thence can pass through them up into the

bed of incandescent coal. Around the opening formed by the contracted part of the shaft there is a pipe *f* through which water is circulated constantly, thereby preventing the edges of the opening from burning out. The top of the shaft is arched over and is pierced by four openings *G*, provided with suitable covers *G'*, for charging in the coal and stoking. In the sides of the ash box, or wind box, are doors *H'* for removing ashes. The blast flues *h* have sliding dampers for regulating the supply of air, which is furnished by an ordinary blower.

At Lasalle, Ill., four of these producers are grouped in a battery, the air supply being taken from a common main, with suitable dampers to regulate the quantity for each producer. Each producer is connected by means of a short flue *I*, through which the gas escapes, with a main flue *B* by means of

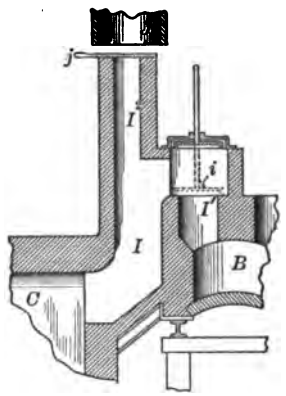


FIG. 211. HEGELER PRODUCER.
Gas connections.

a port *I'*, which can be closed by the valve *i*. The flue I connects also with a chimney *J* by means of a port *I''*, passage of the gas through the latter being prevented ordinarily by means of the slide damper *j*. This arrangement makes it possible to cut any one of the four producers out of the battery while being stoked, fed or repaired, the other three maintaining the supply of gas in the meanwhile. These producers have also been operated at Lasalle by means of induced draught produced by a rotary blower interposed between the battery of four producers and the furnace which they heat. This unique arrangement is described in connection with the Hegeler distillation furnace (vide Chapter X).

Mond Producer.—The gas producer designed by Doctor Ludwig Mond, the eminent English industrial chemist, is illustrated in the accompanying

engraving. This producer is intended specially for recovery of ammonia from the coal consumed, at the same time producing a gas of high calorific value. It is a circular brick shaft, with a sharp bosh at the bottom, built within a cylindrical steel shell. The bottom is water-sealed. The shaft

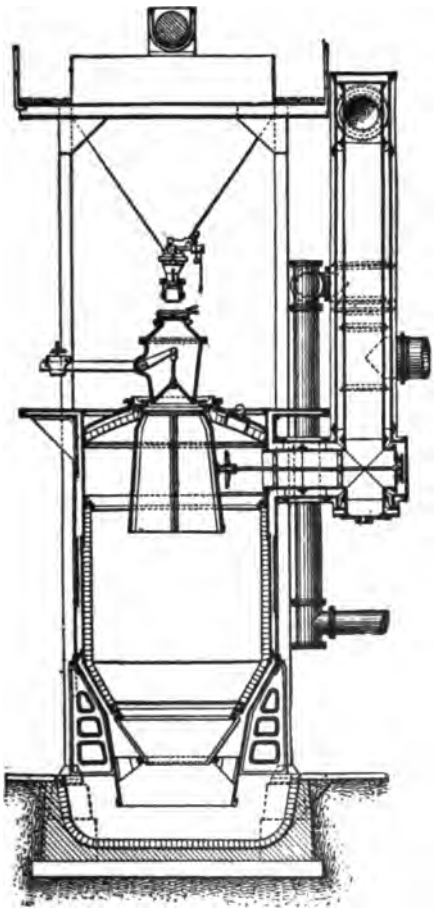


FIG. 212. MOND PRODUCER.

is 9 ft. 8 in. in diameter and 21 ft. high. The top is arched over in dome shape and is provided with a bell and hopper feed. In the engraving a large hopper fed by means of a screw conveyor is shown over the feed hopper of the producer, so that a stock of coal can be carried over each producer and dropped by gravity when it is necessary to feed the producer. This is a

modern arrangement which is to be found in many gas producer installations. At the bottom of the bosh the Mond producer has a conical grate in the center of which there is a round hole through which the ashes descend into the water seal. Beneath the bell and hopper feeding device there is a bell-shaped casting about 7 ft. long, extending down into the shaft, which is kept partially filled with fuel, wherefore the producer is always necessarily full of coal up to the bottom of the aforesaid casting. The external casing of the producer consists of two steel shells. Air saturated with steam is blown into the annular space between them and after being thus distributed and heated passes through the conical grate into the fuel.

In operation the fuel is distilled inside the bell-shaped casing as in an ordinary gas retort. The gas developed forces its way downward through the hot fuel and during its passage the tarry vapors become fixed and give no further trouble. The gas on being taken from the producer at a temperature of 450° to 500° C. is made to pass up and down a series of wrought iron pipes, which are surrounded by annular casings protected from the air by non-conducting material. The steam-saturated air supply for the producer passes through these annular casings and is heated there by the producer gas, whereby a considerable quantity of heat is returned to the producer. After passing through these pipes the gas is conducted through a chamber which is partially filled with water, which is beaten into spray by revolving paddles; the spray washes the dust and soot from the gas. The gas is reduced in this chamber to about 100° C. and is further charged with water vapor from the spray. It passes next through a lead scrubber filled with perforated brick. This scrubber contains sulphuric acid for absorbing the ammonia. In practice it is found best not to use pure acid, but to keep in circulation a solution of ammonium sulphate of about 36% strength, which contains about 2·5% of free acid. At regular intervals a portion of this solution is withdrawn and after recovery of its ammonium sulphate the free acid is pumped back. After the gas has passed through the chambers its temperature is as low as 80° C., but not being fully saturated with moisture, no condensation takes place. It is then passed into a wrought iron scrubber, filled with perforated wood blocks, in which it meets a current of cold water which condenses the steam, the water becoming heated to about 78° C. The gas cooled to 50° C. in this chamber escapes from it ready for combustion. The hot water from the chamber is pumped into a third scrubber of wrought iron, through which a current of cold air is forced. The air is thereby saturated with moisture and heated to about 74° C. and is used as a source of supply for the producer. The water leaves the third scrubber cold enough to be used again in the second scrub-

ber. In this way about half the steam required in the producer is recovered from the gas. Owing to the low temperature at which the producer is worked but little clinker is made and a lining lasts a long time.

The Mond producer gas made from slack coal assaying 11·5% ash, 55% of fixed carbon and 33·5% of volatile matter, including water, in a dry state contained by volume 17% CO_2, 11% CO, 27% H, 42·5% N, 0·4% of olefins and 1·8% of methane. At the works of Brunner, Mond & Co., at Northwich, England, in 1895, there were 10 of these producers in use.[1] They gasify about 23 long tons of coal per 24 hours per producer. In a test with coal containing 7·57% ash, the refuse from the producer averaged 15% C., indicating a loss of 1·33 unit of the carbon in the fuel. The calorific power of the gas obtained was 81·02% of that of the coal. One ton of sulphate was obtained from 23 tons of coal. For each ton of coal gasified, 1583 kg. of steam had to be added to the blast, 0·195 ton of coal being required to produce it. The total extra consumption of coal, including the recovery of the ammonia, was 0·242 ton per ton of coal gasified, i.e., in gasifying 23 tons of coal and recovering the ammonia a total of 28·56 tons was consumed.[2]

The yield of ammonia is the highest when the producer is worked as cool as compatible with proper combustion of the fuel. The most successful results have been obtained by introducing 2·5 tons of steam into the producer for every ton of coal consumed. About 70% of the nitrogen contained in the coal is recovered in the form of ammonium sulphate, the yield of which amounts to 90 lb. to 100 lb. per 2240 lb. of coal. Notwithstanding the special means which are required to remove the water vapor from the gas before the latter is burned and the extra trouble and expense, it is said that these are a good deal more than justified by the recovery of ammonium sulphate, which is no doubt the case. The gas is remarkable for its high percentage of hydrogen and carbon dioxide; notwithstanding the presence of the latter, the calorific power of the gas is high. It is said that when such mixture is passed into the heated chambers of a Siemens furnace the hydrogen reduces a considerable quantity of carbon dioxide to monoxide.[3]

Siemens Producer.—The familiar Siemens gas producer was essentially a fire brick box, oblong or square in plan, the back wall and side walls being vertical and the front wall sloping inward from near the top at an angle of 45° to 60°. The grate was either of the inclined plane type or a combination of step grate and inclined plane grate. This producer was usually built in a pit, the top of the producer being made to correspond with the ground level. The gas was taken off through a chimney communicating

[1] The Mineral Industry, IV, 75. [2] H. A. Humphrey, Proc. Inst. Civ. Eng., CXXIX, 190.
[3] Journal of the Iron and Steel Institute, 1897, Vol. II.

with a horizontal tube and thence passed downward through another vertical leg. In these tubes, which were made of iron, the gas was cooled and its density was thereby increased, so that a draught was produced and the air supply for combustion of the coal was thus drawn into the generator.[1] This type of producer has many defects, which for a long time were lost sight of because of the recognized excellence of the Siemens regenerative system in connection with which the Siemens producers were generally applied. At the present time, however, but few producers of the Siemens type are to be found in America or Great Britain. They were used at the zinc smelteries

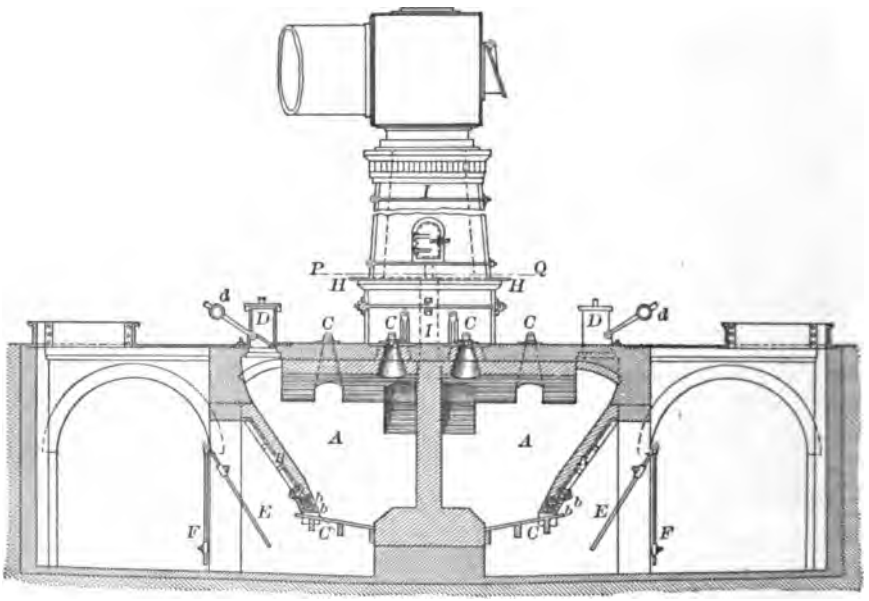


FIG. 213. SIEMENS PRODUCER.
Vertical section.

of the Granby Mining & Smelting Co., at Pittsburg, Kan., and of the Rich Hill Zinc Co., at Rich Hill, Mo., both of which are now dismantled.

There were numerous modifications of the original Siemens producer. In some constructions the plane grate was omitted and the ashes were allowed to rest on a solid hearth, a step grate being placed in the continuation of the inclined front wall near to the hearth. Other producers of the Siemens

[1] The gas would leave the producer at a temperature of 500 to 600° C., and naturally rose in the chimney: in passing through the horizontal and down-coming pipes, which had a superficial area of not less than 60 sq. ft. per producer, it was cooled to approximately 40° C. The draught produced in this manner was weak and the rate of combustion was low; the inefficiency of the device led to its abandonment.

type were made with closed bottoms and blown with a steam jet, a modification to which the Siemens producer easily lent itself. Another modification had curtain walls so built as to effect destruction of the tarry vapors. A circular form of Siemens producer was also designed.

The Siemens type of producer does not permit a bed of coal of sufficient thickness to be carried to allow the degree of combustion which is attained with the modern closed-hearth producers; the grate area is too small, and the cooling of the gas to provide the necessary draught, although it may not lead to loss of heat in connection with regenerative furnaces, was obviously disadvantageous. In the later producers the siphon was omitted, the gas being conveyed from the producer to the place of combustion by as short

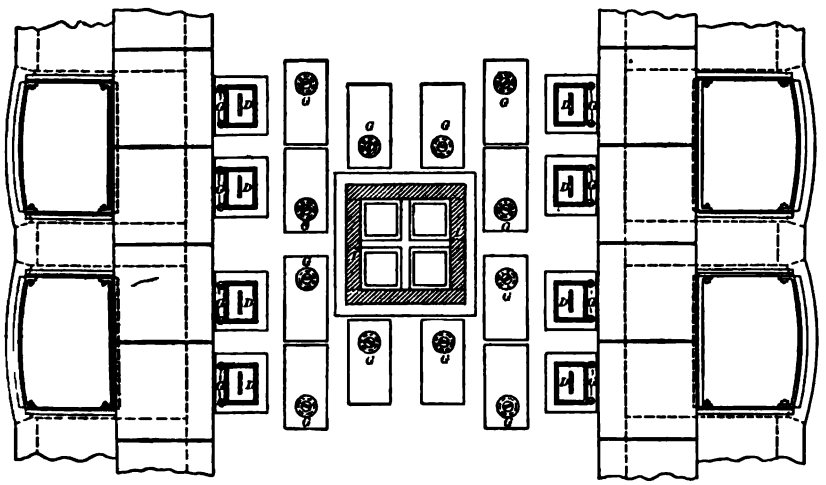


FIG. 214. SIEMENS PRODUCER.
Plan and section on line PQ of Fig. 213.

a flue as possible, so as to retain the initial heat. A battery of Siemens producers, such as was used at Pittsburg, Kan., and at Rich Hill, Mo., are illustrated in the accompanying engravings.

Smythe Producer.—This is a modern American producer which is extensively used in the vicinity of Pittsburg, Pa. It is a cylindrical fire brick shaft, generally 8 ft. in diameter, enclosed by a steel shell. The principal dimensions are given in the accompanying engraving, which shows a producer with an inclined grate. It is claimed that this producer will gasify 1000 lb. of Pennsylvania bituminous coal per hour. It is built by the S. R. Smythe Co., of Pittsburg, Pa., which also builds a water-sealed producer without a grate.

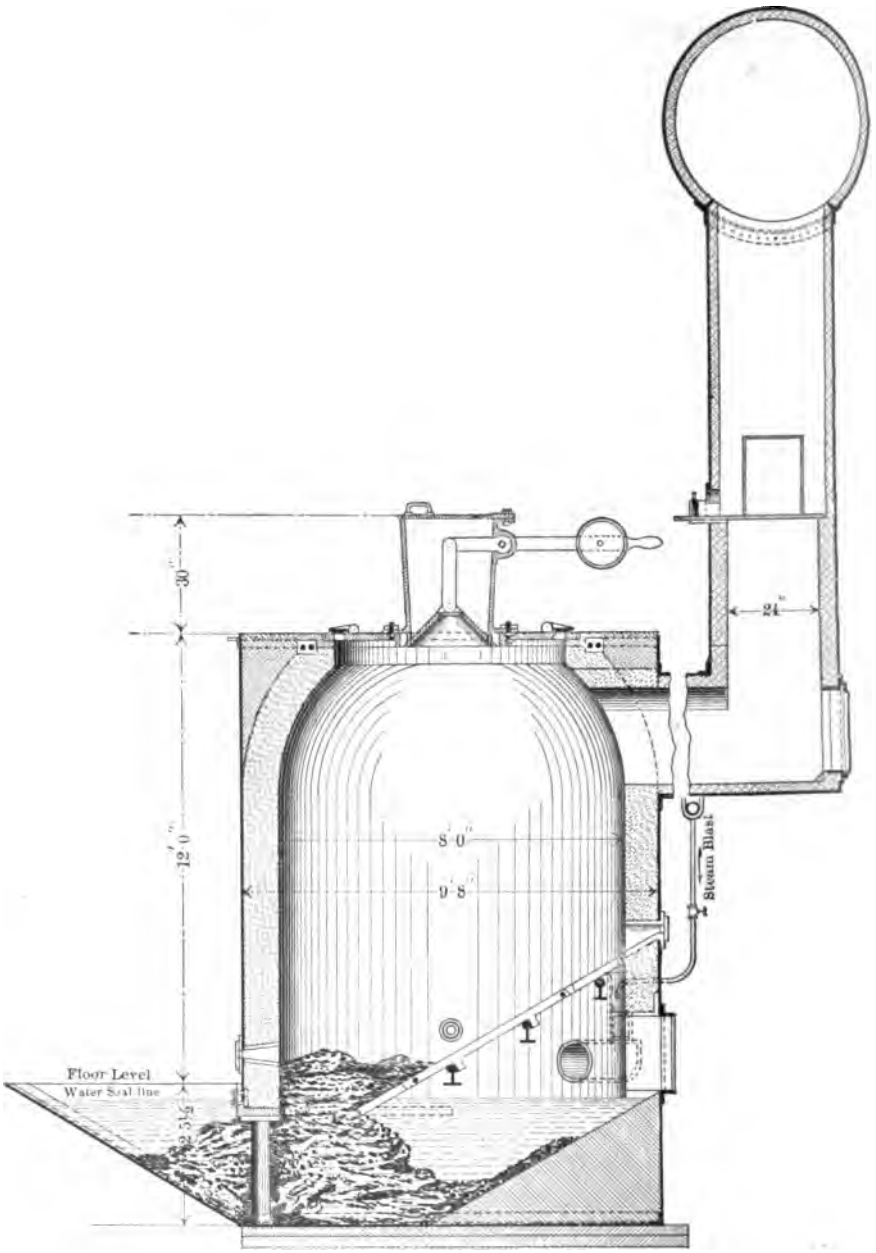
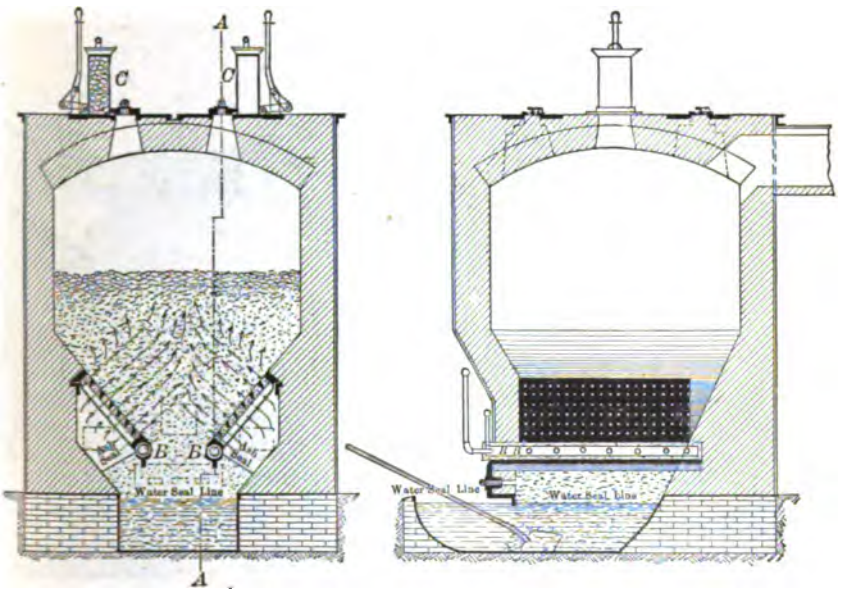


FIG. 215. SMYTHE PRODUCER.

Swindell Producer.—The Swindell producer is a circular fire brick shaft, built up within a cylindrical steel shell. Half way down, the shaft has a sharp bosh and on two sides below the bosh there are inclined rectangular grates, under which the blast of air and steam is introduced. The shaft stands in a basin of water, constituting a water seal, from which the ashes are shoveled out. This water seal, according to the designers, is used only for the advantage gained in removing the ashes and the water surface has been reduced as much as possible in accordance with the idea that a large water surface with steam blast creates a surplus moisture by the continual

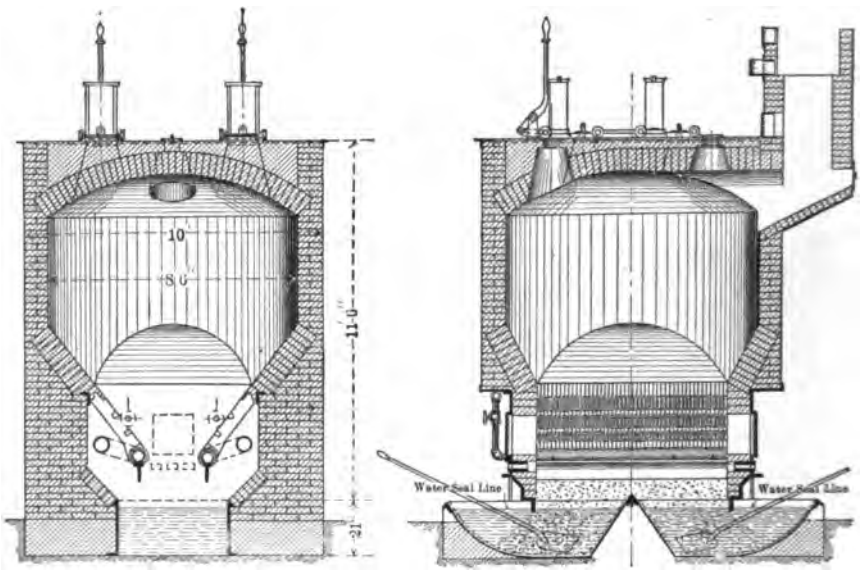


FIGS. 216 AND 217. SWINDELL PRODUCER WITH SINGLE FRONT.
Vertical sections on lines at right angles to each other.

moving of hot ashes in the water seal; and that such moisture must be overcome by heat or allowed to pass through the fire as free steam, which escapes with the non-fixed combustible gas, causing a waste of heat. The producer is blown with steam through the pipes *BB*. The grate is placed about 2 ft. above the level of the water in the basin, in order to allow the ashes to become partly cool before they reach the water, so as to cause as little steam generation as possible.

The top of the shaft is arched over in dome shape and is provided with the necessary poke holes and feed holes. The coal is fed by a special device,

which consists of a coal box combined with a poke hole stopper, all on a sliding plate. In order to feed the producer, the coal box filled with coal is pushed by means of a lever over the opening in the producer, and after dropping its coal is drawn back into position for refilling, the opening in the producer being covered in the meanwhile by the poke-hole stopper, which is in combination with the coal box. This arrangement is gas tight and is not exposed to the action of fire. The Swindell producer is made in two forms, the double front and the single front; it costs about $785 to erect at Pittsburg, Pa., one of the former type, not including foundation, steam connections, or charging platform; one of the latter type costs about $685.



FIGS. 218 AND 219. SWINDELL PRODUCER WITH DOUBLE FRONT.
Vertical sections on lines at right angles with each other.

The producer with a shaft 8 ft. 6 in. in diameter will gasify about eight tons of Pittsburg run of mine coal per 24 hours. This producer is extensively used in the United States, especially in the vicinity of Pittsburg, Pa.

Talbot Producer.—This is a cylindrical brick-lined shaft furnace, with no grate, a central blow pipe and a water-sealed bottom. It is claimed to be able to gasify first-class Pennsylvania bituminous coal at the rate of 800 to 1000 lb. per hour, or 16 to 20 lb. per sq. ft. It is one of the newer designs and embodies various improvements in the charging hopper, poke holes, shutting-off damper, etc. A large number of them are already in use in the United States.

Taylor Producer.—The well known producers designed by Mr. W. J. Taylor and built by R. D. Wood & Co., of Philadelphia, Pa., are made in two forms: (1) the revolving-bottom type and (2) the water-sealed-bottom

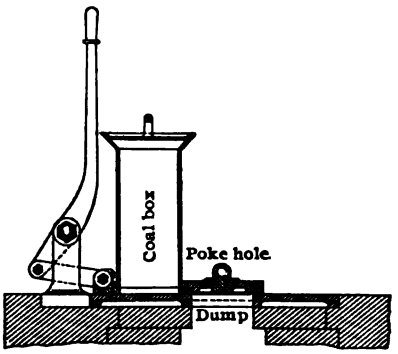


FIG. 220. SWINDELL COAL HOPPER.

type. The producer with revolving bottom is a circular brick-lined shaft contained within a cylindrical steel shell. The brick lining does not extend to the bottom, but is supported on a bearing plate carried by brackets riveted

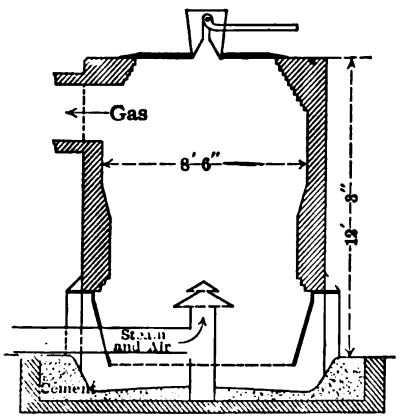


FIG. 221. TALBOT PRODUCER.
Scale, ⅛ in. = 1 ft.

to the shell part way up. Below the line of the bearing plate the shaft is continued for a short distance by means of a bosh of perforated steel plate. In some producers the fire brick lining is not carried down so far, the con-

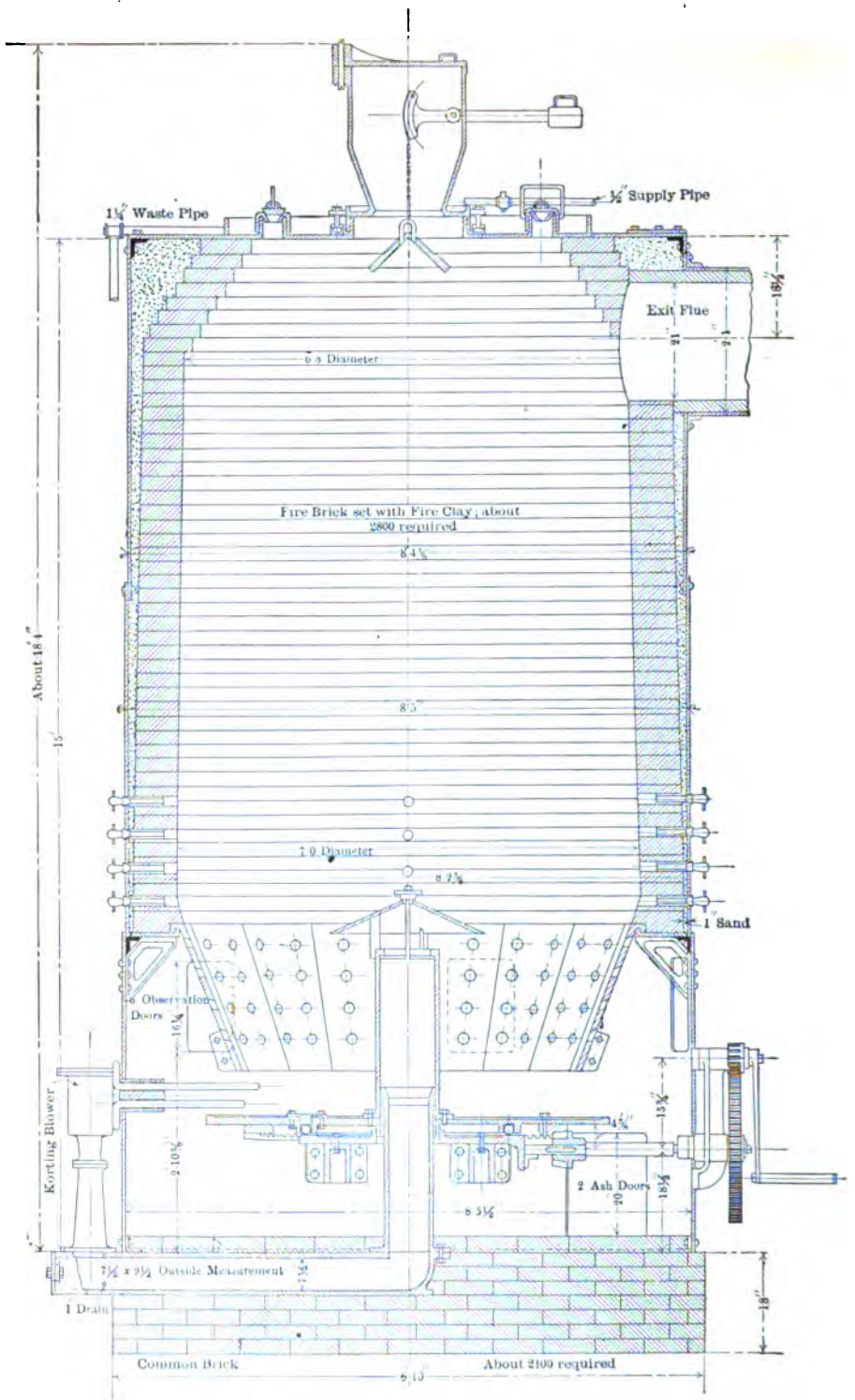


FIG. 222. TAYLOR NO. 7 PRODUCER, WITH REVOLVING BOTTOM.
Scale, ⅜ in. = 1 ft.

tinuation of the shaft being a double shell, between which water is circulated; this type is known as the half water-jacketed. Directly under the bosh there is a horizontal circular plate so arranged that it can be revolved by suitable mechanism. This plate, or revolving bottom, is of greater diameter than the bottom of the bosh and is placed at such a distance therefrom that when it is revolved the ash, which forms its own slope at an angle of about 55°, is discharged over the periphery into the sealed ash pit (which is under blast pressure), this being done without stopping the producer and with little interference with the development of gas. Through the revolving bottom a central air pipe rises vertically into the bosh. In regular operation of the producer the line between the ashes and fuel is kept about 6 in. above the cap on the air pipe, thus permitting the incandescent coal to come into contact only with the brick or water-jacketed lining, while all unprotected iron work is kept away from the heat.

The revolving bottom is turned whenever the ashes rise too far above the desired line, say every six to 24 hours according to the rate of working. The bed of ash is maintained about 3·5 ft. deep on the revolving bottom in the larger sizes of producers, so that ample time is given for any coal which may pass the point of air-admission without being consumed to burn entirely out. It is claimed that tests of a week's duration have been made which have shown a loss in the ash of less than 0·5% of the original carbon in the coal.

The turning of the revolving bottom causes a grinding action in the lower part of the fuel bed and closes any channels that may have been formed by the blast, thus keeping the percentage of carbonic dioxide in the gas at a minimum. A few turns of the crank at frequent intervals will keep the fuel bed in a solid condition, reducing the necessity of frequent poking from above. Ashes are removed once a day, requiring but a short time and interfering but little with the continuous operation of the producer.

The Taylor producers are generally blown by means of a steam jet. A fan blower may be used if more convenient, but then a small steam pipe must be run into the vertical air pipe to supply the steam necessary for preventing clinkers and keeping down the temperature of the producer. The injected air and steam being introduced through a central pipe, from which it is discharged radially, the tendency of the gas to travel next the walls, which is the line of least resistance, is avoided. The central pipe is placed with its top at a point sufficiently high to carry the required bed of ashes, the top of which should never be brought as low as the top of the air pipe. Peep-holes are placed in the walls of the producer so that the dividing line between the ashes and the incandescent coal can be ascertained at any time. In case the dividing line becomes higher on one side than the

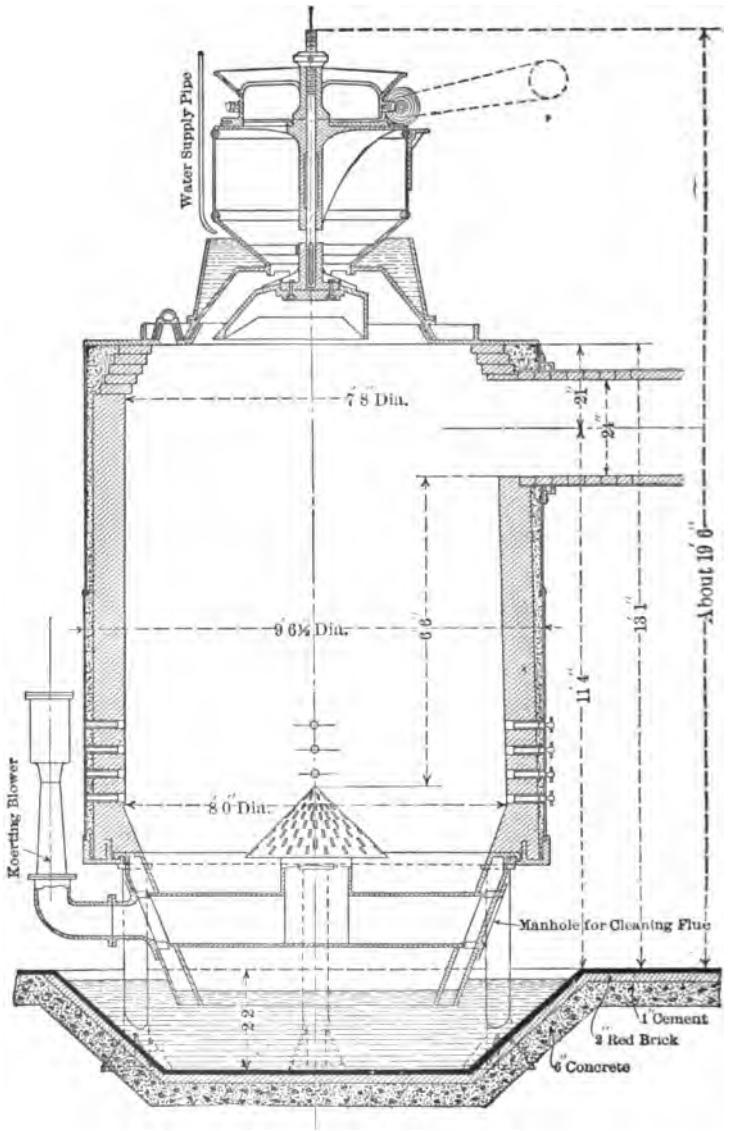


FIG. 223. TAYLOR WATER-SEALED PRODUCER, WITH BILDT FEED.
Scale, ¼ in. = 1 ft.

other, which sometimes happens, four sets of scrapers are arranged just above the revolving table, any of which may be pulled out in case the ashes grind down too fast on one side; this retards the discharge on that side and levels the ash bed.

The perforations in the bosh are for the admission of punch-bars, which are inserted through the observation doors in the lower casing, for the breaking up of occasional clinker which by inattention or bad coal, or both, may have become too large to pass down and out without trouble. The half water-jacketed type of producer is recommended especially for gasifying coal of inferior quality that is likely to clinker, since the clinker will not adhere so readily to the smooth sides of the water jacket as to fire brick, and, moreover, the water jacket is not liable to injury when the poker bars are used from above. In special instances producers with water jackets extending all the way to the top are built, but for calorific considerations any water jacket producer is of course inferior to one with a fire brick lining.

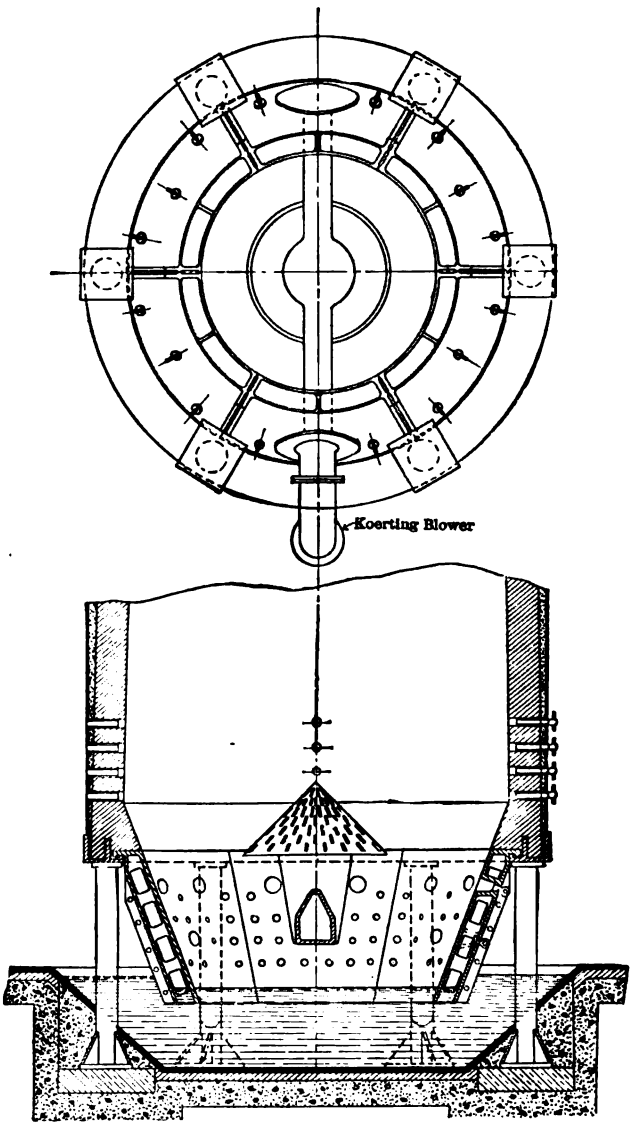
At the top the Taylor producers are arranged with the ordinary bell and hopper feed and poke holes, or with the Bildt continuous automatic feeding device. The Taylor revolving bottom producers are also arranged sometimes with conical ash hoppers at the bottom so that the ashes may be discharged periodically by gravity into a car standing underneath.

The Taylor gas producer with water-seal bottom is similar in general design to the revolving bottom type, but the revolving bottom and its mechanism are omitted and the perforated bosh dips into the basin of water, whence the ashes are shoveled out in the same manner as in other water-sealed bottom producers.

The Taylor producers with revolving bottoms are made regularly in the following sizes:

Size No.	Inside Diam. of Brick Lining or Jacket	Area of Fuel Bed	Height to Top of Casing
8	8 ft.	50·3 sq. ft.	15 ft.
7	7 ft.	38·5 sq. ft.	15 ft.
6	6 ft.	28·3 sq. ft.	15 ft.
5	5 ft.	19·6 sq. ft.	15 ft.
4	4 ft.	12·6 sq. ft.	12 ft.
3	3 ft.	7·0 sq. ft.	10 ft.
2	2 ft.	3·1 sq. ft.	10 ft.

Taylor gas producers are in use at the works of the Illinois Zinc Co., at Peru, Ill., and at those of the New Jersey Zinc Co., at South Bethlehem and Palmerton, Pa., and of the Bertha Mineral Co., at Pulaski, Va.



FIGS. 224 AND 225. TAYLOR WATER-SEALED PRODUCER.

Scale, ¼ in. = 1 ft.

Fig. 221: View from bottom showing bosh plates. Fig. 222: Vertical section of bottom on a plane at right angles to that of Fig. 220. In setting the producer, square holes are left for the columns to stand in; after the columns are set and secured to the mantel rings, the holes are filled with concrete and cemented over.

Wellman Producer.—The Wellman producer is a circular brick shaft built inside of a cylindrical steel shell. For about half way down the shaft increases in diameter, and then is contracted in the form of a bosh. The roof is arched over in dome shape and is provided with the usual poke holes and bell and hopper feed. This producer has a plane grate slightly inclined and is driven by means of a steam jet under the grate. Provision is made for putting in a false grate by pushing bars through the fire above the ashes in such a manner that they act as a grate while the clinkers below are being

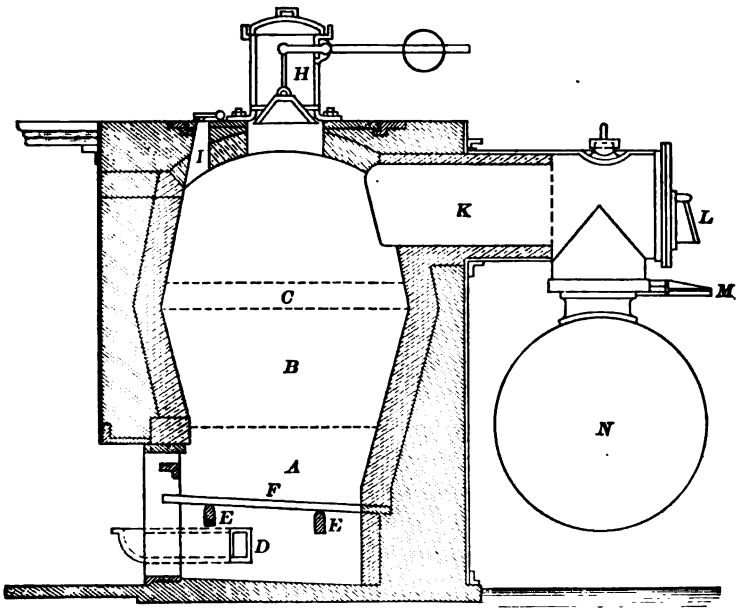


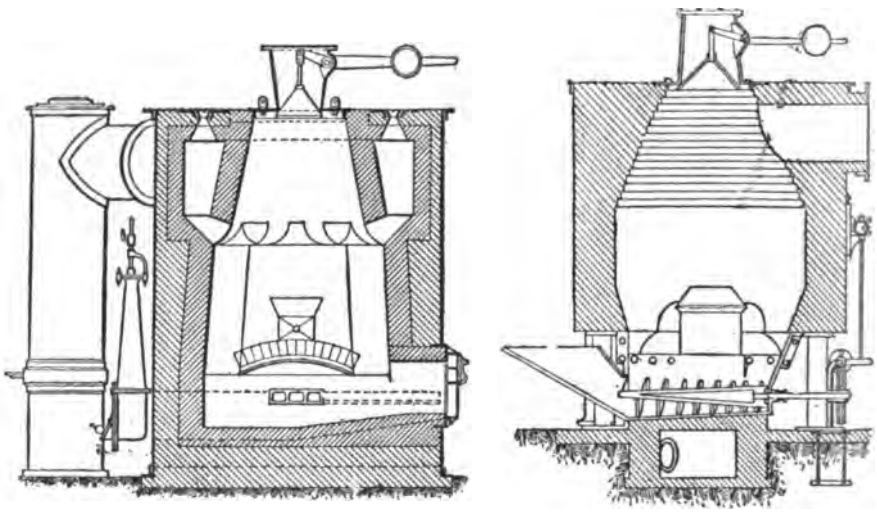
FIG. 226. WELLMAN PRODUCER, VERTICAL SECTION THROUGH CENTER.

A, ash-zone; B, combustion-zone; C, distillation-zone; D, steam-blower; E, bearing-bar; F, grate-bar; H, hopper; I, poke hole; KK, neck to gas main; L, door for cleaning neck; M, damper; N, gas main.

removed. During this operation the fire must be separated from the main receiving tube, else the gas will back down through the fire on the workmen; therefore a separating damper is provided for each producer.

Wilson Producer—This producer, which was patented by Brooke and Wilson in 1877, was one of the first of the modern producers of the shaft-furnace type without a grate. It is still used extensively in Great Britain. It consists essentially of a circular fire brick shaft built within a cylindrical steel shell. In the earlier forms the shaft had no bosh, but on the contrary,

tapered in toward the top. The conical top was surrounded by an annular chamber or gas passage, into which the products of distillation entered through openings in the side of the shaft and traversing the hot coke the tarry matter was completely decomposed. The coal is charged at the top by a bell and hopper feed. Across the bottom of the producer there is a hollow ridge of brickwork, which communicates with the shaft by a series of openings on each side. The mixture of steam and air being blown into this flue escapes into the bed of fuel through those openings. Since the steam and air are thus blown into the middle of the producer the diameter of the shaft must not be so great that the air cannot reach its circumference.



FIGS. 227 AND 228. WILSON PRODUCER.
Fig. 227: Original form. Fig. 228: Mechanical discharge.

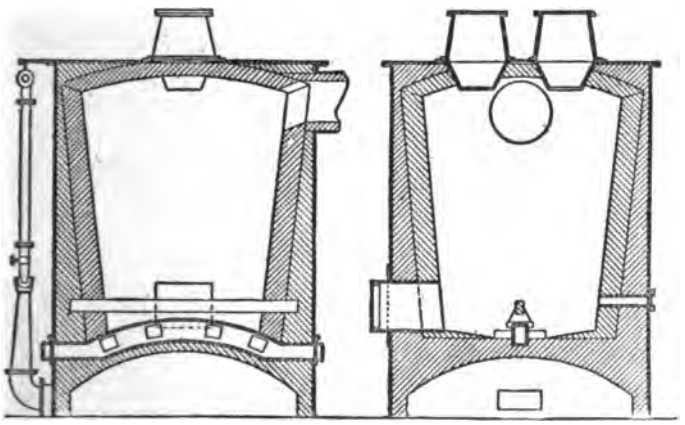
The usual diameter is about 8 ft., but it may be made as great as 12 ft.[1] The ridge in the hearth of the producer divides the latter into two halves, each of which is provided with a cleaning door. When it is necessary to clean out the ashes, which should be done about once in 12 hours, the steam supply is shut off and iron bars are inserted through small side doors so that the ends rest on the brick ridge. The cleaning doors are then opened and the ashes are raked out, after which the doors are closed, the bars removed and the fuel having settled down, steam is turned on. This producer is said to be run ordinarily so as to gasify about 26 lb. of coal per

[1] A. H. Sexton, Fuel and Refractory Material, p. 155. A diameter of 12 ft. would be excessive and hardly to be recommended.

square foot of hearth per hour, but under favorable conditions the rate may be made as high as 40 lb.[1] Both these figures are greatly in excess of what is aimed at in ordinary gas producer practice (vide p. 303).

There is a modified type of the Wilson producer in which the annular gas chamber is omitted and the shaft tapers slightly toward the bottom. The air is supplied through an iron channel connecting with an underground flue and a strong iron bar is fixed across the producer to support the ends of the cleaning bars.

The Wilson automatic producer has a flask-shape shaft and the bottom is formed with two troughs, in each of which works an endless screw. The



FIGS. 229 AND 230. WILSON PRODUCER.
Modified Type.

latter carry the ashes outward and discharge them continuously. The bottom is filled with water, which keeps the screw cool and also serves as a seal to prevent the escape of gas. This producer is built somewhat taller than the ordinary producer and the air and steam are blown into the fuel at a higher level.

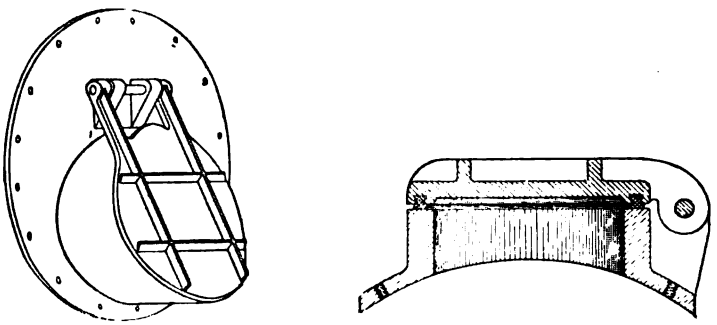
GAS CONNECTIONS.—A furnace heated by an independent producer is connected directly therewith. Zinc distillation furnaces are sometimes so large that a battery of producers is required to serve each one, in which case a single canal will lead from the battery to the furnace. In other gas firing installations a large volume of gas may be required at numerous points and the producers arranged in one battery are caused to discharge into a single main pipe, from which branches are taken off to the various points of consumption.

[1] Sexton, op. cit.

The size of the gas canal is obviously a function of the quantity of gas which is to be conveyed through it. It will depend therefore upon the volume of gas, its temperature, the velocity with which it is to be led through the canal and the length of the canal. When the canal is to be a long one these data should be carefully computed, but in planning short canals, wherein the allowance of an excessive carrying capacity does not add greatly to the first cost, it is generally sufficient to adopt a rough rule, e.g., to make the branch from each producer a pipe of one fourth the diameter of the producer and the main canal equal in carrying capacity to the sum of the branches.

Messrs. R. D. Wood & Co., of Philadelphia, state the following with respect to piping producer gas:

"Connections should be of such size and so designed and constructed as to convey the gas with as little loss of its initial temperature as possible, and



FIGS. 231 AND 232. EXPLOSION DOOR AND HAND-HOLE.

should be provided with suitable valves, safety devices and sufficient hand holes and manholes. The loss in efficiency in piping long distances is greater in bituminous than in anthracite gas. In the former the loss is increased owing to the greater condensation and deposition of the unfixed heavy hydrocarbons, while in the latter (anthracite) practically no loss results except from cooling. Probably 500 ft. is the maximum distance to which bituminous producer gas should be carried; and in such instances it is essential to have the flues of ample diameter—the greater the distance the larger the flue—making allowance, of course, for the partial consumption of the gas along the line. It is usually best to line the flues with fire brick, or other non-conducting material, for their entire length, though cast iron mains of small diameter, 18 in. or less, preferably protected with asbestos on the outside, are used for services which do not justify the outlay for the larger lined mains.

"The size of connection to each producer should be about one quarter the diameter of the producer inside of the lining—thus an 8 ft. producer should have a 24 in. connection. The mains, when reasonably short, should have the same area as the sum of all producer connections feeding them."

The gas main should lead to the furnaces by the shortest possible way, with no contractions and no more bends than are necessary. Sharp turns should be avoided, inasmuch as there is a greater loss of head by friction in one of them than in a considerable length of straight pipe. The necessity for a damper or cut-off valve at each producer has been previously referred to. At intervals of 12 to 15 ft. in the mains there should be handholes, provided with light, water-sealed covers. Since these are to serve as safety valves, as well as openings for cleaning the canal, it is needless to say that the covers should never be fastened down. At the lowest part of the gas canal there should be a tar sump with a water-sealed cover, which will serve both as an explosion safety valve and as an opening for cleaning the canal.

Natural Gas.

Within the last two decades the use of natural gas as a fuel has become of importance in the United States, first in Western Pennsylvania and West Virginia, next in Ohio and Indiana, and most recently in Kansas. The ephemeral character of a natural gas supply, however, has already reduced the industrial importance of this fuel in the vicinity of Pittsburg, Pa., where it has now become a luxury, employed chiefly for domestic purposes, rather than a technical fuel, while in Indiana the supply is also waning rapidly to the great inconvenience of the Indiana works which have relied upon it; in Kansas, however, the natural gas resources have been but lately opened and are not yet seriously diminished. Inasmuch as both in Indiana and in Kansas natural gas is largely used as fuel by zinc smelters a consideration of the principles governing its combustion is important in a treatise on the metallurgy of zinc.

Occurrence of Natural Gas.—Two distinct modes of natural gas occurrence are recognized, which are distinguished by Professor Orton[1] as the shale gas and reservoir gas formations. The former is characterized by wells of comparatively small volume, which lack uniformity of rock pressure and derive their gas from no definite horizon, but have good staying properties, weak flows being maintained for long periods. The latter is characterized by large wells, which draw their supply from a definite stratum of sandstone, conglomerate, or porous dolomite, and give approximately the

[1] Bulletin of the Geological Society of America, X, pp. 99 to 106.

same pressure irrespective of their widely differing volumes. The gas formations of both Indiana and Kansas are of this type. The gas of these porous rocks generally comes to a sudden end, says Professor Orton; oil comes in and fills the pipes, or salt water shuts off the gas like a light blown out by a gust of wind. Only by constant care and attention in removing those substances from the pipes can the life of the well be maintained, especially in its later stages.

The natural tendency of the gas is to accumulate at anticlinal folds or the higher portions of the porous formation in which it occurs, petroleum and salt water being found at the lower levels in the same formation. The theory is that the gas and the two liquids, when coexistent, arrange themselves according to their specific gravities. The pressure under which the gas is stored is known as the rock pressure. There is a difference of opinion as to the cause of this pressure. Professor Orton considers that it is hydrostatic, being due to the head of the water entering the porous formation and in support of this theory showed that the pressure of the wells of the Trenton limestone in Ohio agreed exactly with what would result from a head at the level of Lake Erie, while water followed the gas into the rock as the gas was exhausted. Professor I. C. White has called attention, on the other hand, to the facts that gas exists at points in New York, Pennsylvania and West Virginia at pressure exceeding any possible artesian pressure, while the sands as depleted continue dry. In these cases at least the pressure would appear to be due solely to the causes which produced the great accumulations of gas. Nevertheless it is true that in some fields the life of the wells is suddenly terminated by the influx of water. The appearance of water in a well is not, however, necessarily an indication that it comes from the same stratum as the gas, since it may leak in through defective casing from the overlying strata.

Volume and Pressure.—The so-called rock pressure is that of the gas reservoirs as shown by closed in wells. The open flow of a well is computed from the one minute pressure, which is what is registered after the flowing well has been closed in one minute. The actual flow of a well in use is considerably less than the open flow because it will be discharging against more or less line pressure. The open flow is determined largely by the character of the gas rock or sand. If the latter be loose and open the flow is larger than when it is hard and compact. From this it follows that wells which show the same rock pressure may make a different output, while frequently the smallest open flow may be found with the greatest rock pressure. If the compact condition of the sand be of limited extent, shooting the well may permanently benefit it; otherwise, it will not. A well exhausts the

formation in its immediate vicinity and gradually takes its supply from parts of the field more and more remote, decreasing in rock pressure and open flow in proportion to the distance from which it is drawing. By shutting in and resting a well its rock pressure and open flow may be revived to a degree approaching the limit of the field as a whole. When a single well is overworked water may leak in from the superior rock strata through defective casing, either improperly put in originally or subsequently corroded. There is an economic limit to the pressure under which gas wells should be worked, and a limit to the number of wells that should be drilled on a given area.[1] When a well is not lost by drowning out, and continues dry, the gas of the area it draws from may be almost completely won by pumping after the rock pressure itself becomes too low. In this way gas is still obtainable in some cases where the rock pressure is only a few pounds, but the installation of a pumping plant and its operation adds of course materially to the cost of the gas.

Gradual Exhaustion.—The record of the Indiana field is instructive as to the probable duration of others of similar character. According to J. C. Leach, natural gas surpervisor of the State, in his report to the State Geologist for 1900, the decline in the rock pressure throughout the field was at first gradual and fairly uniform. As the supply of gas decreased the pressure became less uniform, and the rise of the salt water cut off the more elevated portions of the gas strata, thus dividing the field into numerous small gas areas, varying in pressure, the draught on one not affecting the others. The close proximity of the salt water to the gas in the Indiana formation prevents the complete exhaustion of the gas over a large area of the field. The original rock pressure in Indiana was about 325 lb. per sq. in. After 10 years' drain, in 1896 the average pressure was 220 lb.; recently it has been declining at the rate of 20 to 25 lb. per annum, and at the end of 1900 was only about 110 lb. Many of the wells are lost by the invasion of salt water when the pressure falls to 100 lb. The Indiana field comprised an area of 2850 sq. miles and the original pressure was about the same in all parts of it. The Iola field, wherein most of the zinc smelteries of Kansas are situated, is small. The original rock pressure was about 325 lb. The gas was first utilized for zinc smelting in 1896, but until 1899 the drain was not very heavy. At the end of 1902 the pressure had fallen to about 200 lb. as a general average for the field.

Cost of Drilling for Natural Gas and Construction of Pipe Lines.—Natural gas wells are commonly put down by the ordinary method of churn drilling. In Kansas it is customary to tube the wells with wrought

[1] Mineral Resources of the United States, 1900, p. 646.

iron pipe 2 in. or 3 in. in diameter, the casing being 5⅝ in. The cost of drilling is 80c.@ 1·00 per foot. At Iola, where the wells are about 1000 ft. deep, the total cost per well is about $1500 in general. This includes everything, the contractor delivering the well complete and closed in.

In order to prevent the gas of a high pressure well from escaping through the open porous rock near the surface, wherein it would lose its pressure and issue at a distance from the well wherever it might chance to find an outlet, the modern practice of preparing a gas well is to case below the water and tube to the top of the rock where the high pressure gas is found. A packing is placed at that point, whereby all communication with the well above outside of the tube is shut off.

The cost of the pipe line from the wells to the works depends of course

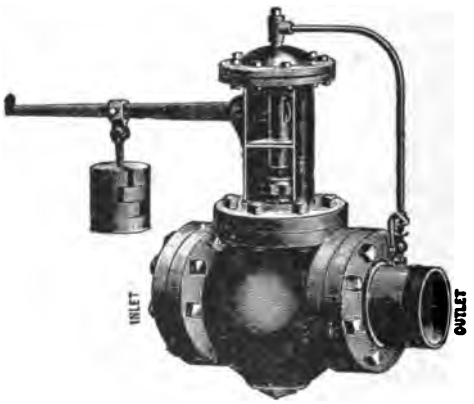


FIG. 233. HIGH PRESSURE OR REDUCING REGULATOR.

upon its length and varies with the market price of the material and the cost of labor in making the installation. The construction of a 6 in. pipe line might be expected to cost about 90c. per foot, reckoning the pipe at 65c. and the labor and incidentals at 25c.; a 4 in. pipe line would probably cost about 50c. a foot, reckoning the pipe alone at 35c. At the zinc smelteries in Kansas it is the common practice to run 4 in. lines from the wells to the works, but lately some 6 in. lines have been put down.

DETERMINATION OF THE VOLUME DELIVERED BY A GAS WELL.—The volume of a natural gas well is commonly estimated from the "one-minute pressure," i.e., the pressure registered by the gas in the well after having been closed in for the period of one minute, or preferably by direct measurement with a Pitot gauge.

In order to determine the output in cubic feet of a gas well by the pres-

sure shown on the gauge after the well has been closed for one minute, the time being noted from when it was commenced to close the gate and pains having been taken to do the latter as quickly as possible, the gauge pressure in pounds is multiplied by 0·07; multiplying the product by the volume in cubic feet in the casing gives the cubic feet per minute, from which the number of cubic feet per hour and per day can be computed by multiplying by 60 and 1440 respectively. For example, suppose the one minute pressure shown by a well 1120 ft. deep tubed with 2 in. pipe were 220 lb., the content of the tube of that length and diameter would be 24·5 cu. ft.

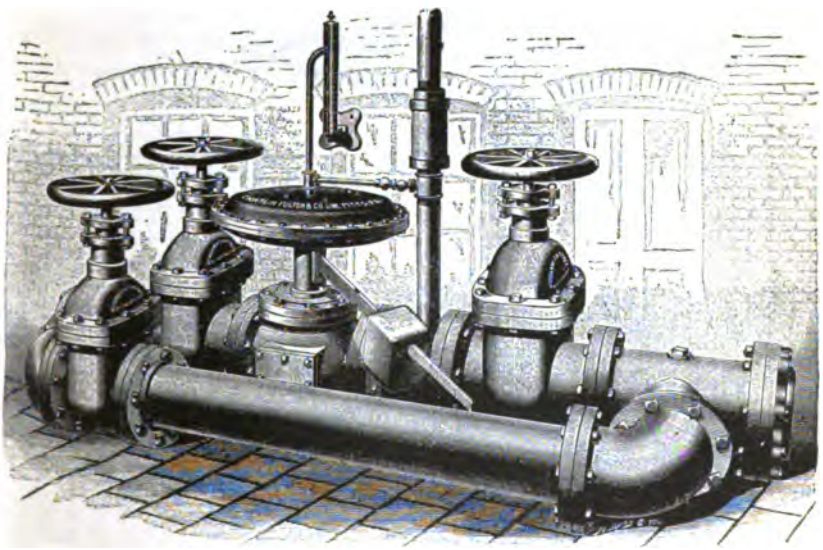


FIG. 234. LOW PRESSURE REGULATOR FITTED WITH BY-PASS.

The output of the well would be therefore $0{\cdot}07\times220\times24{\cdot}5=377{\cdot}3$ cu. ft. per minute $=22{,}638$ cu. ft. per hour $=545{,}312$ cu. ft. per 24 hours.

The flow of gas through pipe lines is determined by meter, by means of the Pitot tube, or by computation from the initial and discharge pressure, size and length of pipe line. The meters directly measure the volume of the gas which passes through bellows of known capacity, filled and emptied automatically. When the volume of gas is so large as to require a very expensive meter to measure the whole flow, the proportionate meter is employed, in which an aliquot portion of the current is diverted and measured.

The flow through a pipe line can be determined very well by means of the

Pitot tube. In its simplest form this is a small metal tube inserted in the pipe with a short leg bent so that its opening is directly opposed to the direction of the current. The pressure caused by the impact of the current is transmitted through the tube to a pressure gauge of any kind, such as a column of water or of mercury, or a Bourdon spring gauge. From the pressure thus indicated and the known density and temperature of the flowing gas is obtained the head corresponding to the pressure, from which the velocity may be calculated.

In a modification of the Pitot tube described by Professor S. W. Robin-

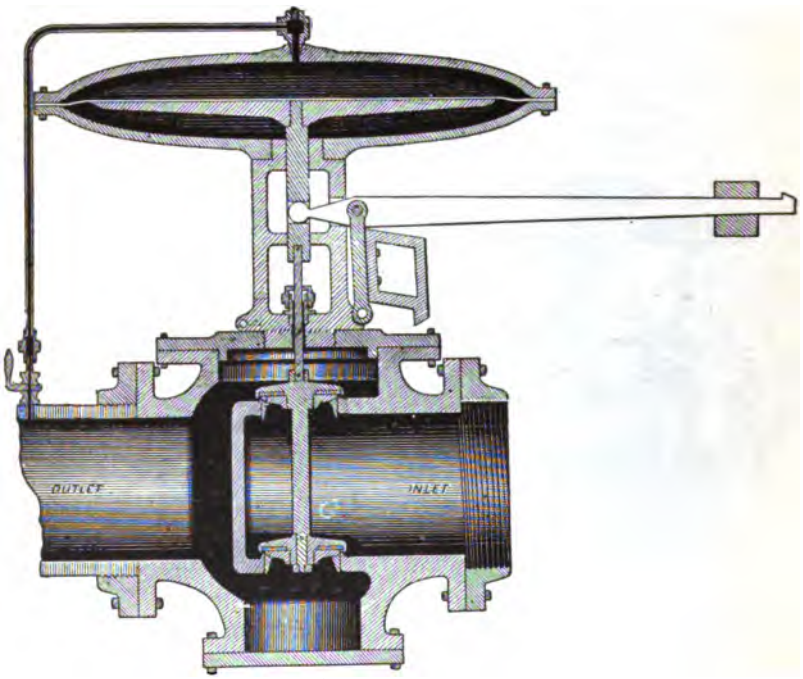


FIG. 235. LOW PRESSURE REGULATOR, WITH AUTOMATIC CUT-OFF.

son[1] there are two tubes inserted into the pipe conveying the gas, one of which has the plane of the orifice at right angles to the current, to receive the static pressure plus the pressure due to impact; the other has the plane of its orifice parallel to the current, so as to receive only the static pressure. These tubes are connected to the legs of a U-tube partly filled with mercury, which then registers the difference in pressure in the two tubes, from which the velocity may be calculated. Comparative tests of Pitot tubes with gas

[1] Report of the Ohio Geological Survey, 1890, VI, pp. 548 to 594.

meters for measurement of the flow of natural gas have shown an agreement within 3%.

PRESSURE REGULATORS.—It is impracticable to introduce natural gas into the furnaces wherein it is to be burned at the pressure at which it issues from the wells; nor is it advisable to bring the gas into the limits of the works at any such pressure. Means have to be adopted consequently to reduce the pressure, which is done commonly by regulators designed upon the principle of reducing valves. In general two regulators, or two sets of regulators are required. These are called the reducing, or high pressure, regulator and the low pressure regulator. The zinc smelters of Kansas who had a supply of gas of about 300 lb. per sq. in. original pressure adopted the practice of reducing it to approximately 50 lb. with a high pressure regulator and thence by means of a low pressure regulator to approximately 6 oz., at which pressure more or less it is burned in the roasting and smelting furnaces.

The regulators in common use are of the type manufactured by the Chaplin-Fulton Manufacturing Co., of Pittsburg, Pa., which are illustrated in the accompanying engravings. The sectional drawing of the low pressure regulator with automatic cut-off shows the mechanism of the apparatus. The design of the high pressure regulator is essentially the same, but the head or diaphragm case is smaller in diameter than for the low pressure regulator. These regulators are made in the following sizes, in connection with which the list prices are given:

1 inch	inlet	with	1,	1¼, 1½ or 2	inch	outlet.....	$20.00
1¼ "	"	"		1¼, 1½ or 2	"	"	25.00
1½ "	"	"		1½, 2 or 2½	"	"	30.00
2 "	"	"	2,	2½, 3 or 4	"	"	40.00
2½ "	"	"		2½, 3 or 4	"	"	55.00
3 "	"	"	3,	3½, 4 or 6	"	"	70.00
4 "	"	"	4,	4½, 5, 5½ or 6	"	"	90.00
6 "	"	"	6,	8	"	"	150.00
8 "	"	"	8,	10	"	"	225.00
10 "	"	"	10		"	"	325.00
12 "	"	"	12		"	"	450.00
16 "	"	"	16		"	"	650.00

There is usually a discount of 25% from the above prices, which are f. o. b. Pittsburg, Pa. The weight of these regulators is approximately 100 lb. to the inch, i.e., a 2 in. regulator weighs 200 lb., and a 6 in. regulator 600 lb. The sizes commonly used by the Kansas zinc smelters are from 4 in. to 6 in. The high pressure regulators are designed to withstand any pressure up to 600 lb., and will reduce to any point between 5 and 150 lb. The low pressure regulators will take gas at 150 lb. and reduce it to any point between that and 4 oz.

Connection of Regulators.—Before connecting the regulator the gas line

should be thoroughly blown out to free it from sticks, stones, dirt and other rubbish. A safety valve should be placed somewhere on the line between the high and low pressure regulators, set to blow off at a few pounds above the pressure carried on the low side. In adjusting the regulators it is important to observe that the small cock leading to the diaphragm is open. The lever and weight are then placed in position and the gas is turned on slowly, weights being added until the pressure gauge on the low side shows the number of pounds or ounces desired. If from any cause the gas leaves the line, the automatic cut-off of the low pressure regulator closes and remains closed until opened by hand. The valves on these regulators are balanced so that variations of pressure in the supply mains do not affect the pressure in the works, an important feature since the operation of the furnaces in which the gas is being burned would be affected by fluctuations in the pressure. Regulators on gas lines and in smelting works generally have a by-pass fitted to them, so that gas can be delivered temporarily around them in case of accident to the regulator, the flow being then regulated by hand.

Another important accessory to a natural gas line is a safety valve on the low pressure line to carry off the surplus gas in case of any accident which might permit the high pressure gas suddenly to enter the low pressure pipes. These valves are made to blow off at any desired number of ounces, so that they can be set a few ounces above the low pressure which is carried. They are entirely closed so that they cannot be tampered with.

Composition of Natural Gas.—Natural gas is essentially a mixture of marsh gas, CH_4, with relatively small quantities of hydrogen, olefiant gas, carbon monoxide, carbon dioxide, oxygen, nitrogen and hydrogen sulphide. The composition of natural gas obtained from various places in the United States is shown in the following table:

Constituent	Fostoria, Ohio *a*	Findlay, Ohio *a*	St. Mary's, Ohio *a*	Muncie, Ind. *a*	Anderson, Ind. *a*	Kokomo, Ind. *a*	Marion, Ind. *a*	Pittsburg Pa. *b*
Hydrogen (H)	1·89%	1·64%	1·94%	2·35%	1·86%	1·42%	1·20%	22·00%
Marsh gas (CH_4)	92·84	93·35	93·85	92·67	93·07	94·16	93·57	67·00
Olefiant gas (C_2H_4)	0·20	0·35	0·20	0·25	0·47	0·30	0·15	1·00
Carbon monoxide (CO)	0·55	0·41	0·44	0·45	0·73	0·55	0·60	0·60
Carbon dioxide (CO_2)	0·20	0·25	0·23	0·25	0·26	0·29	0·30	0·60
Oxygen (O)	0·35	0·39	0·35	0·35	0·42	0·30	0·55	·80
Nitrogen (N)	3·82	3·41	2·98	3·53	3·02	2·80	3·42	3·00
Hydrogen Sulph. (H_2S)	0·15	0·20	0·21	0·15	0·15	0·18	0·20	*c*5·00

a Eng. and Min. Journ., April 21, 1894. *b* Average composition of natural gas in the vicinity of Pittsburg, Pa., according to S. A. Ford, in Trans. Am. Inst Mining Eng., XIV, 433; 100 liters of such gas weigh 64·8585 g. and 1000 cu. ft. are equal in heating power to 55 lb. of the average Pittsburg bituminous coal, i.e., 2000 lb. of coal=36,360 cu. ft. gas. *c* Ethyl hydride (ethane), C_2H_6.

It will be observed from the above analyses that natural gas is the most concentrated fuel that is known in technical practice, being considerably superior to the gas which is distilled from coal, known commonly as illuminating gas, and far superior to the ordinary fuel gas obtained either from anthracite or bituminous coal, which is diluted by a large percentage of incombustible and inert nitrogen. Keeping in mind the chemical composition of natural gas and the concentrated character of its combustible, the principles governing its economical combustion will be readily understood from what has been written previously as to the combustion of coal and producer gas.

Quantity of Air Required for the Burning of Natural Gas.—One cubic meter of producer gas containing 22·8% CO, 8·5% H, 2·4% CH_4, 0·4% C_2H_4, 5·2% CO_2, 0·4% O and 60·3% N, has a calorific value of 1170 calories, wherefore 1000 calories are developed by 0·855 cu. m. of gas, which requires for combustion 0·869 cu. m. of air and produces 1·602 cu. m. of combustion products, comprising 0·267 cu. m. of CO_2, 0·133 cu. m. of H_2O and 1·202 cu. m. of N. One cubic meter of pure marsh gas has a calorific value of 8480 calories, wherefore 1000 calories are developed by 0·118 cu. m. of gas, which requires for combustion 1·126 cu. m. of air and produces 1·244 cu. m. of combustion products, comprising 0·118 cu. m. of CO_2, 0·236 cu. m. of H_2O and 0·890 cu. m. of nitrogen. Whereas for the combustion of 1 cu. m. of producer gas of the composition specified above there is required 1·106 cu. m. of air, the combustion of 1 cu. m. of marsh gas necessitates the supply of 9·512 cu. m. of air. The proportion of air increases with the percentage of ethane in the gas, and the quantity of air required to consume 1000 cu. m. of natural gas of the average composition is slightly less than 11,000 cu. m. In practice, however, it is preferable not to use the full amount necessary for complete combustion, the proportion of 1 : 10 being found to give the best results.[1] It will be observed that in respect to the proportion of air supply practice has determined a radical difference between natural gas firing on the one hand and both producer gas firing and direct firing on the other hand, inasmuch as no excess of air is used with natural gas, while producer gas requires upward of 20% excess of air when the latter is not preheated and ordinary direct firing all the way from 50% to 100% excess.

In natural gas firing it is even more important than in producer gas firing to insure an intimate mixture of the gas and air, since whereas in producer gas firing the air and gas are more or less of equal volumes, in

[1] F. H. Oliphant. Mineral Resources of the United States for 1896, comprised in the eighteenth Annual Report of the United States Geological Survey.

natural gas firing the quantity of air required is in volume ten times as great as the quantity of gas to be burned. Moreover, producer gas and air are of nearly the same specific gravity, the former being approximately 0·97, assuming the latter as unity; while on the other hand the specific gravity of natural gas is only 0·55 to 0·60, air being again assumed as unity.

Dissociation of the Hydrocarbons.—At the temperatures which are frequently obtained in industrial firing, especially in zinc smelting, the hydrocarbons of a fuel gas, either artificial or natural, are subject to dissociation, i.e., they are split up into their elementary constituents, carbon and hydrogen. If this happens in the presence of an adequate supply of air both the hydrogen and the carbon will be burned, the former to aqueous vapor and the latter to carbon dioxide. If, however, the supply of air be insufficient only the hydrogen may be burned, while the carbon will be deposited as soot. With natural gas, of which upward of 90% is hydrocarbon, more difficulty is liable to be experienced on this account than with ordinary fuel gas, which is comparatively poor in hydrocarbons. This dissociation of hydrocarbons with the accompanying deposition of soot is liable to cause trouble in metallurgical firing with natural gas unless it be prevented. The fact that natural gas is dissociated in that manner is, however, one of the attributes to which it owes its great heating power, the latter being due largely to the radiation from the incandescent particles of carbon which are set free. If they be deposited as soot the remedy is simple, consisting merely in providing a requisite supply of air and insuring its thorough mixing with the gas. This was not understood in the early days of natural gas firing at Iola, Kan., and deposits of soot were permitted to form around the retorts in the distillation furnaces, reducing the heat conductivity of the walls of the retorts and consequently the extraction of metal from the ore, the soot sometimes accumulating to the extent that the passages for the combustion products between the retorts would be completely closed and had to be opened by barring. Since 1899 a better knowledge of the method of burning natural gas in the Iola district has entirely eliminated this difficulty.

In the early history of the use of natural gas in the vicinity of Pittsburg, Pa., the gas was discharged into the furnaces directly from simple pipes, which method was found of course to be defective inasmuch as the mixture of gas and air was slow and imperfect, just as it is when a basin of kerosene oil is ignited in the open air. Also the gas was admitted to the furnace under too high a pressure, keeping the stream consolidated and preventing access of air. The effect of this is observed in an exaggerated degree in the case of wells of natural gas at high pressure when it has been impossible to close in the gas and the latter has been ignited; combustion does not take

place under those circumstances until the gas has reached an altitude of many feet, where its pressure has diminished and it has become scattered so that the air gains access to it.

The method of burning natural gas in furnaces was improved by dividing the flow through a number of small openings, each entangling a portion of air before the gas is ignited, instead of causing the gas to issue directly out of the pipe or through a perforated end. The greater the number of the divisions and the lower the pressure up to a certain limit, the more perfect was the combustion. Frequently the gas supply pipes terminate in mixers which draw in air on the principle of the well known Bunsen burner and give good results. In good practice in natural gas firing the gas is now rarely admitted into the furnace at a pressure exceeding 6 oz. According to F. H. Oliphant[1] the best results have been secured from a burner with a series of small openings, $\frac{1}{8}$ to $\frac{5}{32}$ in. diameter or less, under a pressure not greater than 4 to 6 oz. The gas is conveyed through a tube 1·5 in. diameter and 16 to 18 in. long, surrounding each jet of gas, with a proper opening at the bottom of the tube to admit air on the principle of the Bunsen burner.

Relative Efficiency of Natural Gas and Coal.—A natural gas of the composition 2·18% H, 92·6% CH_4, 0·31% C_2H_4, 0·5% CO, 0·26% CO_2, 3·61% N and 0·34% O, has a calorific power of approximately 1,100,000 B. T. U. per 1000 cu. ft. One ton of Pittsburg, Pa., coal containing 55% fixed carbon and 32% volatile combustible has a calorific power of 28,750,000 B. T. U. Consequently 2000 lb. of the average Pittsburg bituminous coal is theoretically equivalent to 26,000 cu. ft. of natural gas of the above composition, which is more or less typical of the natural gas of Western Pennsylvania, West Virginia, Ohio and Indiana, as will be seen by comparison with the analyses given in a previous section. The theoretical calorific power of natural gas cannot of course be attained in practice, but according to F. H. Oliphant 1000 cu. ft. will evaporate under favorable conditions 1000 lb. of water at 212° F., equivalent to a utilization of 966,000 B. T. U.

Practically the relative efficiency of natural gas and bituminous coal is different from that which is calculated theoretically. As to what is the true practical equivalent authorities differ, as would be naturally expected, since the conditions under which the tests have been made are far from identical. Assuming that 1 cu. ft. of natural gas will actually evaporate 1 lb. of water and that 10 lb. of water can be evaporated with 1 lb. of coal, which we know can actually be accomplished with good Pittsburg bituminous coal, 1 lb. of coal would be equal to 10 cu. ft. of natural gas, whence 2000 lb. coal=20,000 cu. ft. of natural gas. According to S. A. Ford 1000 cu. ft.

[1] The Mineral Industry, X, 464.

of the natural gas obtained in the vicinity of Pittsburg, Pa., is equivalent in heating power to 55 lb. of the average Pittsburg bituminous coal, i.e., 2000 lb. of coal=36,360 cu. ft. of gas. This means the theoretical equivalent. According to Barr in his treatise on *Boilers and Furnaces*, tests in making steam in boilers at Pittsburg, Pa., showed that 1000 cu. ft. of natural gas was equivalent to 80 to 133 lb. of bituminous coal, according to the quality of the coal and the gas and the skill in firing, these figures being equivalent to ratios of 2000 lb.=25,000 cu. ft. and 2000 lb.=15,000 cu. ft.

The variation in the above figures is due of course to the difference in the character of the gases and the coals, especially the latter, and the conditions under which they were burned and their calorific values utilized. A clearer understanding of the true basis for comparison is to be gained from a consideration of the properties of the pure fuels. Methane (CH_4), or marsh gas, when burned completely, has a calorific value of 23,513 B. T. U. per pound. One cubic foot weighs 0·04463 lb. Consequently the calorific value of a cubic foot is 1049 B. T. U.; of 1000 cu. ft. it is 1,049,000 B. T. U. A pure coal consisting of 93·76% C. and 6·24% H. would have a theoretical heating value of 17,556 B. T. U. per pound, or 35,112,000 per 2000 lb. The potential energy in 2000 lb. of such coal would be equivalent therefore to that of 33,470 cu. ft. of methane.

The combustion of 1000 cu. ft. of methane, or 44·63 lb. would result in 122·73 lb. CO_2+100·42 lb. H_2O+596·26 lb. N=819·41 lb. total gas. The combustion of 59·75 lb. of coal, containing 93·76% C and 6·24% H, which has the same potential heat effect as 44·63 lb. of methane, would result in 205·59 lb. CO_2+33·57 lb. H_2O+599·93 lb. N=839·09 lb. total gas. It will be observed that although there is no great difference in the quantity of the combustion products there is a great variation in their composition. While that from the methane contains about 15% CO_2 and 12·25% H_2O, by weight, that from the coal contains 24·5% CO_2 and only 4% H_2O. Consequently the specific heat of the former is 0·268 against 0·247 for the latter. If the two fuels be burned in a smelting furnace from which the products of combustion escape at a temperature of say 2372° F. (1300° C.) the available heating effect, leaving aside losses by radiation, will be the theoretical minus what is carried away as latent and physical heat in the products of combustion. In the case of the methane the heat carried away by the combustion products will be:

Latent heat of 100·42 lb. H_2O	96,976	B. T. U.
Physical heat: 819·41×(2372−62)×0·268	507,280	B. T. U.
Total	604,256	B. T. U.

In the case of the coal the corresponding figures will be:

Latent heat of 33·57 lb. H_2O.....	32,419	B. T. U.
Physical heat: 839·09×(2372−62)×0·247....	478,760	B. T. U.
Total	511,179	B. T. U.

The available heating effect from the methane would therefore be 1,049,000−604,256=444,744 B. T. U., while that of the coal would be 1,049,000 −511,179=537,821 B. T. U. This presupposes, however, the combustion of the coal without excess of air, which in practice is not attainable unless it be converted into gas and the latter be burned with highly preheated air. The burning of 59·75 lb. of coal of the composition specified requires 780 lb. of air. If 10% excess be used the escaping products of combustion will be increased by 78 lb., which will carry away 78×0·2375×(2372−62)=42,793 B. T. U. If 50% excess be used the additional quantity will be 213,965 B. T. U. and the available heating effect will be only 1,049,000−(511,179+ 213,965)=323,856 B. T. U. Instead of having to burn 33,470 cu. ft. of methane to obtain the same practical result as from 2000 lb. of pure coal, it would be necessary to burn only about 24,400 cu. ft. If the methane were 95% of the natural gas and the remaining 5% were incombustible, the quantity of natural gas required would be approximately 25,700 cu. ft. If the coal contained 87% combustible (say 55% fixed carbon and 32% hydrocarbons) the natural gas equivalent of a ton would be 0·87×25,700=22,359 cu. ft.

It appears therefore that in the complete combustion of quantities of methane and coal of equal heat potentiality, the character of the combustion products of the former cause more heat to be wasted than in the case of the latter. The relative proportion of such waste is affected moreover by the temperature. Whereas at 2372° F. (1300° C.) the available heat is 444,744 B. T. U. in the case of the methane, against 537,821 B. T. U. in the case of the coal, the ratio being 1 : 1·21, with a chimney temperature of 572° F. (300° C.), the corresponding figures are 840,027 and 910,881, giving a ratio of 1 : 1·08.

The calorific value of natural gas is increased by the substitution of ethylene (olefiant gas), C_2H_4, for part of the methane, and more so by the substitution of ethane, C_2H_6, the values of those gases being 1667 and 1863 B. T. U. per cu. ft. respectively. The practical equivalence between natural gas and coal depends therefore upon the composition of each and especially the manner in which burned. The coal which was assumed in the preceding calculations is approximately the composition of Pittsburg bituminous, leav-

ing out of account its oxygen, nitrogen and sulphur contents, which have a slight effect on the calorific power. Natural gas is commonly burned without excess of air and combustion is nearly complete. Coal is burned with a large excess of air and combustion is seldom complete. The smaller the excess of air and the more complete the combustion the greater the number of cubic feet of natural gas required to equal a ton of a given coal; the greater the excess of air and the less complete the combustion the smaller the number of cubic feet of natural gas required. The largest ratio is to be expected therefore when the coal is converted into producer gas, with the minimum loss of initial energy and the producer gas is burned under conditions which necessitate only a slight excess of air. Comparisons between natural gas and coal on the basis of their theoretical heating powers are of little value unless the conditions of combustion are taken into account.

A comparison between natural gas and other kinds of coal than the Pittsburg bituminous can be made on the basis of the efficiency of the latter. Assuming that 1 lb. of Pittsburg bituminous coal containing 90% of combustible, 8% of ash and 2% moisture has a theoretical heating value of 13,500 B. T. U., and that a pound of Western bituminous coal containing 75% of combustible, 15% of ash and 10% moisture has a theoretical value of 10,150 B. T. U., the relative theoretical efficiency of the two coals will be in the same ratio; practically, however, because of the difficulty in effecting so complete combustion of the inferior Western coal its efficiency will be found considerably lower, and it is probably unsafe to reckon a ton of the bituminous coal of Southern Illinois and Kansas as being equivalent to more than two thirds of a ton of the average coal of Western Pennsylvania and West Virginia, comparing run of mine with run of mine.

Use in Regenerative Furnaces.—The high calorific power of natural gas renders it possible to attain very high temperatures without preheating either the gas or the air. If, however, economy in gas be an important consideration, just as much advantage is to be derived from preheating the air as in the case of producer gas firing. This can be done by the employment of the Siemens system with two regenerative chambers (for air alone) or any of the counter-current recuperative systems. With proper management, even the ordinary Siemens system with four regenerative chambers, two for air and two for gas, may be employed, the dimensions of the chambers being modified of course to correspond to the relative volume of gas and air, which in the case of natural gas is very different from what it is in the case of producer gas. Numerous attempts to pass natural gas through the checkerwork of the ordinary Siemens furnace proved unsuccessful, owing to the deposition of soot from the dissociation of the hydrocarbons, which choked

up the passages, but although it is frequently stated that natural gas cannot be used in connection with regenerative furnaces, because of the rapidity with which the checker-work is filled up with such carbon deposits, Mr. George W. Goetz has pointed out that the deposition of carbon can be prevented by admitting a little steam with the gas before it enters the checker-work.[1]

To let carbon deposited in the checker-work burn out when reversing the furnace is bad practice. Whether a gas deposits carbon or not can be easily seen by having an observation hole (closed by glass) for the checker-work. If carbon is deposited, the moisture necessary to remove it in the form of water gas is easily determined; thus it is consumed in the furnace and is not wasted to the chimney. Care must be taken, of course, that no large excess of steam is given. In the case of coal there are few grades of that fuel which do not give more moisture to the gas than is necessary.

Prevention of Waste in the Use of Natural Gas.—In burning natural gas economy has been but little considered, the supply at Pittsburg (Pa.) and vicinity having become largely exhausted before it was realized that the end was in sight, while a similar carelessness was displayed in Indiana and is being displayed in Kansas. If natural gas were really a costless fuel, of which the supply is limitless, as it has been too generally regarded, there would be obviously no necessity of being economical in its use, the only consideration being to regulate the temperature of the furnace as desired. If on the other hand it be aimed to preserve the supply of gas as long as possible it is essential to attain the desired temperature with the minimum consumption of gas and avoid all unnecessary waste. This is unfortunately lost sight of, with blindness to the common interest, in many districts of which the very welfare is based on the natural gas supply, but in the Eastern States where the cheaper gas has already been exhausted and the more remote fields have passed into the hands of gas-selling companies a wiser practice has become possible.

The steps toward economy in the use of natural gas which have been taken in the vicinity of Pittsburg, Pa., have been in the direction of preventing waste, by stopping leakages in the lines, and by selling the gas by measurement instead of by the orifice basis, thereby compelling the consumer to be more careful and economical. According to F. H. Oliphant it is probable that by virtue of such economies only 1 cu. ft. of gas is now consumed to accomplish the same results that required 3 to 4 cu. ft. for several years after the use of natural gas was first introduced. The appliances for holding back the gas in the wells when it is not required, the

[1] Trans. Am. Inst. Min. Eng. XXII, 682.

packing of the lines at night, when the consumption is lightest, so as to meet the pull in the daytime, the more judicious selection of the position of regulators feeding from the high to the low system, the proportioning of the sizes of the low pressure lines to the demand, the proper location of drips, the installation of the gas compressor where the rock pressure in the wells is no longer able to force the gas through the original pipe line, the introduction of improved joints for connecting the pipe lines and a more thorough stopping of all leaks, no matter how small, are some of the more recent conditions brought about by the decreased volume of the Eastern natural gas fields.

Coal Dust Firing.

The combustion of coal in the form of dust has recently attracted considerable attention and promises to become of importance in the future. For that reason it may be appropriately referred to briefly, although the method has not yet found permanent application in zinc smelting or in any other branch of metallurgy, at least not in the United States. The idea of coal dust firing is not a new one, powdered fuel having been used in the Crompton rotary puddling furnace at the Woolwich Arsenal in England in 1873,[1] and for heating zinc distillation furnaces by Daehne and Hauzeur at an earlier date.[2] The modern application of the art has come from Germany, however, where experiments made in 1893 showed that pulverized fuel could be burned without smoke and with high economy. An experiment in boiler firing by this method made in New York about three years ago did not prove economically successful, although there was no smoke and presumably a complete combustion.[3] Dust firing has recently been applied successfully in the United States, however, in the rotary kilns used in the manufacture of cement.

In dust firing the fuel instead of being introduced into the fire box in the ordinary manner is first reduced to a powder by any suitable pulverizing machine. The dust is blown directly into the combustion chamber of the furnace by means of an air injector similar in construction to those used in oil-burning furnaces or by means of a rotary wire brush (Schwartzkopff system). In either case the apparatus which throws a constant stream of the fuel into the chamber is so located that it will scatter the powder throughout

[1] Journal of the Iron and Steel Institute, 1873, p. 91.

[2] Daehne's dust-fired furnace (Schüttofen) was tried in England and at Letmathe in Westphalia but did not prove successful as may be readily understood, the coal-dust being showered over the retorts (Berg.- u. Hüttenm. Ztg., 1868, p. 766). Hauzeur's furnace (patented Dec. 2, 1871) was better designed and is said to have been successful in Spain (Kerl, Grundriss der Metallhüttenkunde, p. 456).

[3] William Kent, The Mineral Industry, VIII, 134.

the whole space. A furnace is put into operation by first raising its lining to a high temperature by means of an open fire, after which the introduction of the powder is begun, and having been once ignited by the heat radiated from the walls of the furnace, the combustion continues in a regular manner under the action of the current of air which carries in the dust. Dust firing therefore is analogous in principle to gas firing, its advantage over direct firing lying in the intimate mixture which is effected between the very finely ground particles of fuel and the supply of air which can be regulated with precision. The chief objections to the method are the cost of grinding the coal and the high degree of skill required for the management of the furnace so as to obtain the proper proportions of air and coal. It

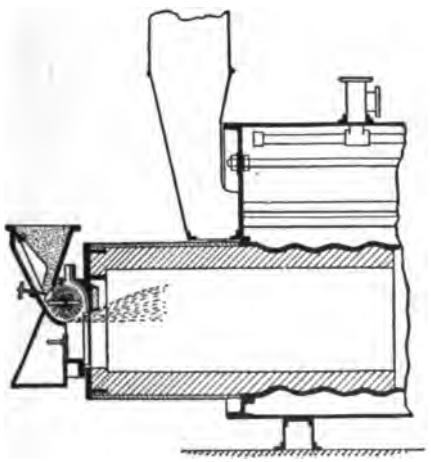


FIG. 236. SCHWARTZKOPFF COAL DUST FIRING DEVICE.

would appear, however, that no more skill would be required in this method of firing than in gas firing.

According to the *American Manufacturer* of December 13, 1900, in order to obtain the best results from dust firing the coal must be reduced cheaply to a very fine powder and must be of a strictly uniform grade. The powder mixed with air must be carried in an unbroken stream into the combustion chamber. The air current must be so regulated that it will hold the coal powder in suspension within the furnace until complete combustion is effected. A sufficiently high temperature must be continuously maintained in the furnace to insure perfect combustion of the powder. The advantages of the method are summarized as follows: (1) economical and complete combustion of the fuel in a manner similar to gas firing; (2) saving in

labor; (3) adaptability and ease of regulation to meet any requirements; (4) decreased wear and tear of furnaces; (5) saving of time in starting up furnaces and rapid stoppage of firing in case of necessity; (6) intimate contact of the fuel with air, whereby the minimum excess over the theoretical volume is employed and waste of heat is thus avoided.

According to a United States Consular Report, published in May, 1901, many experiments have been made recently in Germany to fire furnaces with coal dust, principally with apparatus having some form of fan blower to introduce the fuel into the furnace. Not much success has attended the application of these devices. The Schwartzkopff apparatus has, however, met with favor, owing to the facility it affords for utilizing slack or low grade coal. A German Imperial Commission appointed to test smoke-consuming appliances made trials of this apparatus, and the report was favorable. As to the conditions of the process, a highly heated fire chamber is necessary for the ignition of the coal dust. Contact with the boiler walls interferes with ignition, so that in fitting up a boiler for coal dust firing a fire chamber lined with refractory material has to be provided, because only such a chamber can be kept constantly at the required high temperature. This is most easily accomplished with flue boilers when the flues are lined for a length of from 5 to 8 ft. with fireproof material. Metallurgical firings do not offer any difficulty to the arranging of the fire chamber, because the whole furnace possesses the high temperature required.

IX.

CHIMNEYS, HEAT RECUPERATION AND FURNACE DESIGN.

The general principles governing the methods of supplying air for combustion, furnace design and the recuperation of heat from the waste gases are the same in direct firing, gas firing and dust firing, and though modifications are naturally necessary to meet the particular conditions of each case those subjects are best considered together.

CHIMNEYS.—The functions of a chimney are to create a draught which will afford the necessary supply of air for the combustion of a desired quantity of coal, and to disperse the products of combustion in such a way that they will do no harm. The latter function is performed by the same conditions which are required to effect the former. The draught producing power of a chimney is based upon the effect of gravity in establishing an equilibrium between the column of hot and correspondingly light gas enclosed within its shaft, and a column of the same height of the colder and heavier external air. Obviously therefore the factors which determine the force of draught produced by a chimney are: (1) the acceleration due to gravity; (2) the difference in temperature between the hot gas and the external air; and (3) the height of the chimney.

Theoretical Velocity.—The relative effect of the several factors is shown in the following formula, which gives the theoretical mean velocity in feet per second of the air entering the chimney, neglecting the loss due to contraction at the entrance:

$$V = \sqrt{2\,gh\left(1 - \frac{t'}{t''}\right)}$$

in which

V=velocity in feet per second.
g=acceleration due to gravity=32.
h=height of chimney in feet.
t'=absolute temperature of external air.
t''=absolute temperature of the hot gas.

The expression $\sqrt{2\,gh}=8\sqrt{h}$. If the units of meters and seconds be used, $g=9{\cdot}8$ and $2g=19{\cdot}6$.

Substituting the value of $8=\sqrt{2g}$ in the first equation, we have:

$$V = 8\sqrt{h\left(1-\frac{t'}{t''}\right)}$$

From the above equation it is evident that the theoretical mean velocity of the air entering the chimney varies as the square root of the height of the latter.

The velocity of the hot gases is further proportional to their volumes and to their absolute temperatures, wherefore the theoretical speed in the chimney is expressed by the formula[1]:

$$V' = V\frac{t''}{t'} = 8\frac{t''}{t'}\sqrt{h\left(1-\frac{t'}{t''}\right)}$$

Another formula is expressed as follows:

$$V = \sqrt{\frac{gha\,(T'-T}{1+aT}}$$

in which a is the coefficient of expansion of gases (0·002035 per degree F.) and T' and T are respectively the temperatures of the gas inside the chimney and the external air, the other references being as before. This formula gives somewhat different results from the preceding.

Practically the theoretical velocity can never be attained in a chimney. The shape of the shaft, whether circular or rectangular, tapering or uniform in cross-section, the condition of the brick work (tight or leaky), the character of the interior surface (causing more or less friction), the condition of the atmosphere and many other uncertain circumstances combine to affect the draught. In the case of direct firing the direction from which the wind blows has a very decided effect upon the draught; if it blows

[1] This formula, which is given by F. E. Idell in an essay on the Theory of Chimney Draughts (published as an appendix to Armstrong's Chimneys for Furnaces and Steam Boilers), is derived as follows: Assuming a chimney of 1 sq. ft. area, of h height, the difference in weight in pounds of the column of hot air and a corresponding column of air at atmospheric temperature is

$$d=h(w-w')$$

in which, w is the weight in pounds of a cubic foot of the external air and w' that of a cubic foot of the hot gas. The height, or head, in feet of the column of external air producing this pressure by its own weight is found by dividing the difference in weights by the weight of a cubic foot of air, thus

$$h' = \frac{d}{w} = \frac{h(w-w')}{w} = h\left(1-\frac{w'}{w}\right).$$

The densities or weights of the air and hot gas, under the same pressure, being inversely proportional to their absolute temperatures, $w' : w :: t' : t''$, in which t' is the absolute temperature of the external air and t'' that of the hot gases.

$$h'-h\left(1-\frac{w'}{w}\right)-h\left(1-\frac{t'}{t''}\right).$$

The theoretical mean velocity in feet per second of the air entering the chimney will correspond therefore to

$$V=\sqrt{2\,gh'}=8\sqrt{h}=8\sqrt{h\left(1-\frac{t'}{t''}\right)}$$

The absolute zero is 459·2° below the zero of the Fahrenheit scale and 273° below that of the centigrade scale. Absolute temperatures are computed, therefore, by adding those figures respectively to the direct readings of the Fahrenheit and centigrade thermometers.

strongly into the ash pit the draught may be increased; if it blows in the opposite direction the draught may be lessened. The resistance offered to the passage of air through the grate and fuel, the length, section and changes in direction of the flues are all important considerations.[1]

Coefficient of Retardation.—The effect of the varying factors which tend to diminish the draught of a chimney must be summarized and introduced into the equation representing the velocity of the hot gases in the form of a coefficient, *c*. That coefficient is necessarily so variable that great judgment must be employed in selecting the value to suit a particular case. Because of this uncertainty it is a common practice among metallurgists to rely solely upon the results of previous experience and waste no time upon chimney calculations. However this is apt to result in furnace installations which will not afford the maximum economy in fuel and there is no doubt that the theoretical principles can be advantageously given a greater application; what is needed is more data as to the relation between the theoretical draught of chimneys and the practical results under many different conditions.

F. E. Idell in the appendix to Armstrong's essay on *Chimneys for Furnaces and Steam Boilers*[2] states that data collected from existing chimneys (in direct firing practice) point to a value of the coefficient of friction, etc., between 0·1 and 0·25; in his calculations he assumes 0·2 for ordinary practice. Dr. Ure determined the actual velocity of the hot gases in an iron chimney of 5 in. diameter and 45 ft. height, leading directly from a stove burning charcoal, to be between 0·2 and 0·167 of the theoretical. With respect to the value of this coefficient a distinction must be made always as to whether it is to be applied to the entire firing installation or only to the chimney itself. The values given above refer obviously to the former case. For the chimney alone Peclet, upon whose investigations most of our ideas as to chimney draught are based, considered that the value of *c* would be between 0·38 and 0·85. Giesker makes $c=0{\cdot}33$. Toldt uses the value 0·33 in his calculations, i.e., for the actual draught of the chimney alone, but for the draught in regenerative (Siemens) gas firing systems (worked only by chimney draught) he reckons only one sixth as much, thus making $c=0{\cdot}055$, and prepared his elaboration of the Prechtl table on that basis. The effect of long flues in diminishing the draught of a chimney is remarked also by

[1] In a chimney 50 ft. high, with a temperature of 17° C. for the external air and 290° C. for the hot gases, the absolute temperatures being 290° and 573° respectively, the theoretical velocity in feet per second would be according to the first formula

$$V'' = 8\tfrac{1}{10}\sqrt{50\,(1-\tfrac{290}{573})} = 78{\cdot}5 \text{ ft.}$$

If the coefficient of retardation be assumed 0·2 the actual velocity will be 15·7 ft.

[2] Third American Edition, 1894, published by D. Van Nostrand & Co., New York.

Wm. Kent, who found that in the case of a battery of boilers connected to a chimney 150 ft. high there was a draught equivalent to 0·75 in. of water at the boiler nearest the chimney and only 0·25 in. at the boiler furthest away; the first boiler was wasting fuel from too high temperature of the chimney gases, having too large a grate surface for the draught, and the last boiler was working below its rated capacity, and with poor economy, on account of insufficient draught.[1] Determinations of the diminution of draught in roasting furnace flues are reported by Doctor M. W. Iles.[2]

Geo. A. Davis reports determinations of the actual velocity in 16 chimneys of various dimensions and operating conditions in Great Britain and compares the results with the theoretical calculations.[3] Unfortunately, he does not give the needful data for a close analysis. Many of the examples cited show extremely low velocities, indicating retardation in the flues or furnaces rather than in the chimney itself. However, out of the 16 chimneys 10 show velocities upward of 0·1 of the calculated and five show 0·2 or more. One example is very instructive. This was a chimney 3×3 ft. in section and 100 ft. high. Long, narrow flues ran into this chimney from every portion of the works, new plant being added, from time to time, until it was manifest that no more air would enter at the furnace ends. There is no record of what the velocity in the chimney was at this time. A new chimney having been decided upon, together with a main flue 5×4 ft., into which the shortened narrower flues were to be collected, the new flue (being completed in advance of the new chimney) was connected with the old one, and so much improved its draught that the new chimney was unnecessary. After the new flue and old chimney had been in working order for several weeks, it was determined that the temperature of the escaping gases was 516° F. and the velocity in the chimney 41 ft. per second, which was nearly 55% of the theoretical as calculated by the second formula given above.

In modern gas firing installations, wherein the producer is blown with steam and air or with air alone, the proportioning of the chimney is simpler than in plants wherein the coal is burned directly. The supply of air required for combustion of the coal is furnished positively and in a manner which is completely under control, wherefore that function is removed from the chimney. The air for the secondary combustion may also be advantageously blown into the furnace, either directly or through the recuperative chambers, and this practice is frequently adopted. All that the chimney is required to do is to remove the burned gases from the combustion cham-

[1] Mechanical Engineer's Pocket Book, first edition, p. 734.

[2] Lead Smelting, p. 151 et seq.

[3] Handbook of Chemical Engineering, II, 82.

ber. The excess of air required for the combustion being also much smaller, the volume of the gas to be discharged is known more definitely. Calculation of the actual velocity which will be realized is consequently simplified, the value of the coefficient, *c*, having to be assumed in correspondence only with the friction in the chimney and the flues leading thereto.

Quantity of Gas Discharged.—The quantity of gas that a chimney will discharge is represented by the formula:

in which $$Q=AV,$$

Q = cubic feet discharged per second.
A = area of top of chimney in square feet.
V = velocity in feet per second.

In all these calculations it must be borne in mind that the volume of hot gases increases in the direct ratio of the absolute temperatures. It should be noted that in the above formula V is the actual velocity realized in the chimney and not the theoretical velocity due to gravity.

Uselessness of Very High Chimneys.—It will be observed from the formulæ representing the theoretical draught of a chimney that the velocity of flow varies as the square root of the height of the chimney, wherefore in order to double the velocity it is necessary to quadruple the height. Practically, however, the friction increases so much with the height that it would be necessary to do more than quadruple the latter in order to double the draught in that manner. Chimneys above 150 ft. in height are very costly and their increased cost is rarely justified by the increased efficiency. Very tall chimneys have been characterized by one writer as "monuments to the folly of their builders."

High Temperature of Gas Unnecessary.—Similarly it will be observed from the formulæ that although the temperature of the products of combustion delivered to a chimney are a factor in determining the draught of the latter there is but little to be gained from an increase, entirely apart from the loss of heat that may be experienced, while beyond a certain point there is an actual reduction of discharge. Toldt points out[1] that if it were to be accepted that t″ might increase to infinity, such would not be the case with the discharge because the density of the gas decreases as t″ increases. The expression $\sqrt{\frac{t''-t}{1+at''}}$, in which a is the coefficient of expansion of air, has therefore a maximum value which may be determined by differentiation with reference to t′=0 as follows:

$$1+at''=2a\,(t''-t') \text{ and } t''=2t'+\tfrac{1}{a}.$$

$$a=0{\cdot}003665 \quad \tfrac{1}{a}=273.$$

If t′ = 0°C., t″ = 273°C.

[1] Regenerativ-Gasöfen, p. 212.

Computing the value of $\sqrt{\frac{t''-t'}{1+at'}}$ for different temperatures the following results are obtained:

100° C	= 212° F	= 7·40	$= \sqrt{\frac{t''-t'}{1+at'}}$
200	= 392	= 8·16	"
273	= 523·4	= 8·255	"
300	= 572	= 8·251	"
400	= 752	= 8·110	"
500	= 932	= 7·89	"
∞	= ∞	= 0	"
0	=	= 0	"

From this it appears that the draught of a chimney is already active at 100° C. (212° F.) difference between the internal and external temperatures; that its maximum is attained at 273° C. (523·4° F.), and that there is no significant increase over 200° C. (392° F.).

Expressing the results of the above analysis in different words, the action of a chimney does not depend so much on the velocity produced as on the weight of the products of combustion discharged. The higher the temperature of the products of combustion entering the chimney the greater is their volume and with equal velocity the less weight of gas will actually pass through the same chimney. The velocity it is true will increase with the difference between the internal and external temperatures, but only in proportion to the square roots, while the relative weight of the gases decreases in direct proportion to the temperatures by 1/273 of the original volume for each degree on the centigrade scale. There must therefore be a limit where the action of the chimney reaches its maximum, which, as Toldt has calculated, is attained when the difference of temperature amounts to 273° C., or in other words when the external air is at the mean temperature and the chimney gases are at 300° C.

It may be laid down therefore as safe guides in chimney design, based on the theoretical conditions, that the height of a chimney should not be more than 100 to 130 ft. (approximately 30 to 40 m.) and certainly not more than 150 ft. unless it be necessary to disperse noxious vapor very high in the atmosphere; and that the mean temperature of the gas in the chimney need not be more than 200° C. (392° F.), while anything in excess of 300° C. (572° F.) is useless.

Measurement of Draught Power.—The draught power of a chimney is sometimes measured by inches of water. This may be computed from the following formula:

$$d = h\left(\frac{7{\cdot}61}{t'} - \frac{7{\cdot}95}{t''}\right)$$

in which

d = draught in inches of water.
h = height of chimney in feet.
t' = absolute temperature of the external air.
t'' = absolute temperature of the chimney gas.

This formula refers to temperatures expressed on the Fahrenheit scale.

The design and construction of large chimneys, involving questions of foundation and stability, is too big a subject to be discussed in a treatise of this character. It will suffice to remark that the cost increases greatly with the height; that self-supporting steel chimneys lined with brick are cheaper than ordinary brick chimneys, and that the Custodis method of construction with large radial brick is claimed to be the cheapest of all. The great chimney of the Orford Copper Co., at Constable Hook, N. J., erected in 1900, is of the last type of construction; this chimney is about 350 ft. high, and is exceeded in height by only one or two others in the United States.

Practical Data.—The velocity of the gas rising through a chimney is generally between 10 and 23 ft. (3 to 7 m.) per second. According to Gruner, the maximum for temperatures between 200 and 300° C. is 5 m. (16·5 ft.) when there is no excessive friction, but is only 3 to 4 m. (10 to 13 ft.) when long flues of small cross-section have to be traversed. At higher temperatures the velocity may attain 10 m. (33 ft.). According to Toldt,[1] the practical velocity lies between 3 to 7 m. (10 to 23 ft.) and seldom exceeds the latter limit. The chimneys examined by Davis, previously referred to, showed velocities of 1·4 to 41 ft., the latter being observed in the chimney 100 ft. high, of which the experience was noted.

The volume of the gas that passes through a chimney is influenced by the sectional area of the latter and the velocity of the gas. The sectional area of a chimney remaining the same, its height must be increased according to the rapidity of the combustion that it is desired to effect, the character of the fuel and its thickness on the grate, the resistance to which the current of gas is subjected in passing from the grate to the chimney, and somewhat in inverse ratio to the temperature of the gas up to the limit of about 300° C. According to W. C. Roberts-Austen, [2] with furnaces in which the combustion is not rapid and in which there is but slight frictional resistance, as in those from which the gases pass directly into the chimney, the height of the latter need not exceed 33 ft.; furnaces in which combustion is rapid require a chimney at least 50 ft. high, when they discharge directly into the latter. Other authorities lay down the rule that no chimney should be less than 52 ft. (16 m.) in height. Long reverberatory roasting

[1] Regenerativ-Gasöfen, p. 221.
[2] Introduction to the Study of Metallurgy, fourth edition, p. 317.

furnaces (Fortschaufelungsöfen) should have a chimney 65 ft. high, when the latter is directly connected with the furnace or only a short flue intervenes. For steam boilers chimneys should be 65 to 80 ft. high because of the greater cooling of their gases and the increased frictional resistance. If the flues leading to the chimney are long and especially if they are narrow and crooked the height must be increased up to perhaps 130 ft.

When a chimney is entered by more than one flue it is essential that the various currents of gas should pass into the shaft in parallel directions, so as to prevent suction being checked by the impact on each other of converging streams of gas. It is customary therefore where two flues enter a chimney to divide the latter by a vertical wall sufficiently high to turn the two currents upward before they unite. For some unexplained reason a flue should not enter a chimney at its very base, but at least three or four feet above the latter so as to leave an air cushion below the entering stream of gas. When a chimney draws gas from a series of furnaces of which any one is likely to be out of operation at some time, there should be means to cut off each one completely so that cold air will not be drawn in through it; hence every branch flue should be provided with a tightly closing damper.

Relation between Chimney Section and Grate Area.—The bed of coal resting upon the grate of a fireplace naturally impedes the passage of air due to the suction of a chimney, and it is obvious that there must be a relation between the chimney section and the grate area, or what is the same thing, the "free area" between the pieces of fuel. Upon this relation, in fact, a good deal depends in the practical combustion of coal. The smaller is the free area the quicker is the motion of air through it and the more perfect the combustion, provided that the reduction in suction is due to a diminution of the grate area and not to clogging of the fuel, which by increasing the friction at once diminishes the action of the chimney, since the volume of air supplied is necessarily lessened. For this reason small grates give rapid combustion and large grates slow combustion with the same chimney. Rapid combustion, which is necessarily attended with an excessive supply of air, promotes the production of carbon dioxide, while slow combustion, employing a deficient supply of air, favors the production of carbon monoxide. As a rule therefore for the production of high temperatures rapid combustion on a small grate is preferable to slow combustion on a large grate. The temperature that will be attained in the laboratory of the furnace will depend upon the relation between the size of the latter and the grate area. Thus, as Gruner expresses it, for a given fuel burned in a certain way, the heat developed will be simply proportionate to the weight of the fuel burned, while the temperature will depend on the rapidity of combustion, i.e., on the ratio

between the volume of the zone of combustion and the weight of fuel per hour; the smaller this ratio the higher will be the temperature.

In determining the proper size of grate for metallurgical furnaces, a good deal depends upon the character of the coal. With poor coal, more pounds must be burned to develop the same quantity of heat than with good coal, while moreover the greater quantity of ash and clinker from the poor coal tends to clog the grate and diminish the free spaces through which the air may pass. Therefore a proportionately larger grate area should be provided for poor coal than for good coal, and the ratio between the chimney and the grate should be smaller. The thickness of the bed of fuel is also an important consideration. The more caking the coal, the less of it can be burned on a given grate area; "of very caking coals not more than 12 to 14 lb. per sq. ft. should be burned per hour; if less caking from 14 to 16 lb.; and if non-bituminous, from 16 to 20 lb. may be used." [1] Rankine gives the rate of combustion on certain grates as follows: Slowest rate of combustion in Cornish boilers, 4 lb. per sq. ft. per hour; ordinary rate in such boilers, 10 lb.; ordinary rate in factory boilers, 12 to 16 lb.; ordinary rate in marine boilers, 16 to 24 lb.; quickest rate of combustion, the supply of air coming through the grate only, 20 to 23 lb.; quickest rate of complete combustion of caking coal, with air holes above the fuel to the extent of one thirtieth of the grate area, 34 to 37 lb.; locomotives, 40 to 120 lb. Gruner gives: Furnaces for roasting sulphides, 3 to 8 lb.; fires for stationary boilers, 8 to 20 lb.; reverberatory furnaces for smelting lead, 12 to 16 lb.; furnaces for copper smelting, 15 to 30 lb.; puddling furnaces, 20 to 30 lb.; steel melting furnaces, 41 to 81 lb.; locomotives, 81 to 102 lb. Howe states[2] that in reverberatory copper smelting furnaces the consumption of coal sometimes falls as low as 16 lb. per sq. ft. of grate per hour, and sometimes rises to 36 lb., but about 25 lb. appears to be the best practice. In blende roasting with good coal a combustion of 5 to 7·5 lb. per sq. ft. of grate per hour appears to be required, but under unfavorable conditions 10 or 12 lb. may be necessary. In direct fired distillation furnaces the rate appears to be generally 25 to 40 lb. per hour, but with poor coal and imperfect combustion it may run as high as 50 lb. per hour.

In the case of the moderately low chimneys, 50 to 60 ft. in height, which are directly connected with metallurgical furnaces, the sectional area should be proportioned to the quantity of coal that is to be burned. According to Howe,[3] chimneys in general should be provided with not less than 1 sq. ft.

[1] Sexton, Fuel and Refractory Materials, p. 215. [2] Copper Smelting, p. 29.
[3] Op. cit. p. 35.

of section per 125 lb. of coal to be burned per hour, while in some cases the ratio should be as small as 1 sq. ft. to 75 lb., and in extreme cases (as for example where the chimney is likely to be choked by accretions) even as low as 1 sq. ft. to 50 lb.; for copper smelting furnaces the ratio should not greatly exceed 1 sq. ft. to 75 lb. At 25 lb. per sq. ft. of grate per hour, this would correspond to a ratio of 1:3 between chimney and grate. At Argo, Colo., the ratio between chimney and grate in the furnaces of 1878 was 1:2·98, the consumption of coal being 18·5 lb. per sq. ft. of grate per hour; in 1894 the ratio was 1:2·03 and the rate of combustion 34·6 lb.[1] Direct fired zinc distillation furnaces are generally provided with chimneys of one fourth to one half the grate area. In general the ratio between chimney and grate in metallurgical furnaces is between 1:2 and 1:5. Reiche gives 1:4 as the minimum for a chimney when stone coal is burned, and 1:6 in the case of brown coal.[2] In the case of steam boilers, where the rate of combustion is lower, ratios of 1:6 to 1:8 are common. The still lower rate in roasting furnaces would imply the possibility of a proportionately smaller chimney, but in this case allowance has to be made for the reduction of the draught through the fireplace by the frequent opening of the working doors in the sides of the furnace, admitting a great volume of excess air.[3] Peters prescribes[4] a chimney 3·5×3·5×65 ft. for two reverberatory furnaces, 16×64 ft., each having a grate 2×6·25 ft., which gives a ratio of about 1:4 between chimney and grate, the latter burning about 6·4 lb. of slack coal per sq. ft. per hour.

MECHANICAL DRAUGHT.—The tendency in the most advanced steam boiler practice is already strongly pronounced in favor of creating the necessary draught for the combustion of the coal by mechanical means instead of by a chimney, and it is probable that before the passage of many decades the tall chimneys of the present day will remain only as the monuments of an antiquated practice. The advantages of mechanical draught are numerous. The draught power that is easily obtainable is far in excess of what can be got economically from chimneys. It permits the interposition of "fuel economizers" or heat recuperators to an extent that would not be possible with the comparatively feeble draught of the most powerful chimney because of the obstruction thereto which they would present. The air supply for the coal

[1] Peters, Modern Copper Smelting, p. 445.

[2] Professor H. B. Gale gives the rule that the sectional area of the chimney in square feet should be equal to the number of pounds of fuel to be burned per minute, this corresponding to a velocity of 7 to 11 ft. per second for the chimney gases (W. W. Christie, Chimney Design and Theory, p. 15). According to this rule, the ratio of chimney to grate for rates of combustion of 15, 25 and 30 lb. per sq. ft. per hour would be respectively 1:4, 1:2·4 and 1:2. It will be observed, however, that these figures correspond to a very moderate velocity.

[3] Vide p. 50.

[4] Op. cit., p. 188.

and the rate of travel and escape of the hot products of combustion are completely under control, wherefore more economical combustion can be effected. The capacity of the plant is elastic, i.e., it is capable of a considerable range of variation, permitting the addition or subtraction of grate area without affecting the ultimate efficiency, while on the other hand the capacity of a plant operated by a chimney is rigid, that is, there is a limit to the quantity of coal which can be burned, the necessity for burning an additional quantity involving the erection of another chimney. In operation the mechanical draught system is also the cheaper, inasmuch as the power expended, measured in heat units necessary to produce the same results, may under ordinary conditions be only about 1/75 of that necessary with a chimney, providing there be a proper system of heat recuperation. Finally the first cost of the plant is less.

The most approved system of mechanical draught is the use of large exhaust fans which draw the products of combustion through a heat economizer and discharge them through a steel chimney of sufficient height to disperse them in such a way that they will not be a nuisance.

In metallurgy mechanical draught has been employed in lead smelting for the operation of furnaces of which the products of combustion contain metallic fume which it is profitable to recover by filtering through cotton or woolen bags, the gases having previously been cooled by radiation of their heat in the passage through long sheet iron flues below the point where there is danger of igniting the bags. In the metallurgy of zinc mechanical draught is employed as an essential feature in the production of zinc white direct from ores and there are other classes of furnaces in connection with which it may no doubt be used advantageously. In the manufacture of sulphuric acid by the chamber process mechanical draught is employed with great advantage.

Heat Recuperation.—The great step in advance which was made in the introduction of the Siemens system of gas firing was not in the producers, but in the scientific and well developed system of recuperating heat from the waste products of combustion. The Siemens system continues in use at the present time in substantially its original form. To a less extent countercurrent recuperators are employed, which are also an old invention, the principle dating back to the time of Gaillard & Haillot, Lencauchez, Ponsard, Charneau and Nehse. All these are systems of true heat recuperation, i.e., they recover it from gases which would otherwise waste it, and do not abstract it from the fire box or combustion chamber of the furnace, although the mere transference of heat in that manner, as exemplified in the well known Boëtius furnace, may be highly advantageous.

The terms "regenerative furnaces" and "recuperative furnaces" are commonly employed to designate different types, the former being applied to the Siemens system and the latter to the continuous, or counter-current, system; it is generally safe to infer that such a distinction is made when the two expressions are used in metallurgical literature, but not always. More exactness is desirable. The terms "heat regeneration" and "regenerative furnaces" are misnomers. Regeneration implies a re-creation of heat, which does not take place in such a furnace, the heat wasted from the combustion chamber being simply restored thereto. Recuperation, or recovery, expresses the precise meaning as to what is effected in both types of furnaces. The

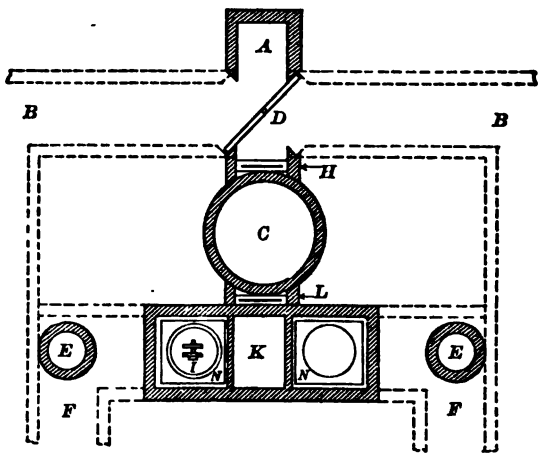


FIG. 237. REVERSING VALVES FOR SIEMENS REGENERATIVE FURNACE.

Horizontal section on line AB of Fig. 238.

A, air inlet; BB, air chambers; C, stack; D, air reversing valve; EE, gas inlets; FF, gas chambers; H, stack damper for air; I, stack reversing valve for gas; K, flue for reversing valve to stack; L, stack damper for gas; NN, water cooled valve seats.

two systems can be appropriately and exactly designated as the "reversing-recuperation" and "continuous-recuperation" of heat.

Siemens System.—In the Siemens system of heat recuperation the hot products of combustion are made to pass through chambers filled with fire brick in the form of a checker-work, to which they impart a large portion of their heat. In the meanwhile the air and gas for combustion enter the furnace through a similar pair of chambers filled with brick checker-work. After a certain time, say 30 minutes, the direction of the gases is reversed. The products of combustion are then caused to pass out through the two cold chambers, while the air and gas enter through the two which have become highly heated, the gas passing through one and the air through

the other. The respective chambers for gas and air are sometimes made of the same size, but more commonly are designed according to the relative volumes of the gas and air and their heat absorbing capacities. An arrangement of flues and valves by which the reversal of the gas currents is effected is shown diagramatically in the accompanying engravings from a paper by H. H. Campbell.[1]

The extent to which heat may be recuperated by the Siemens system depends upon the temperature of the combustion products discharged into

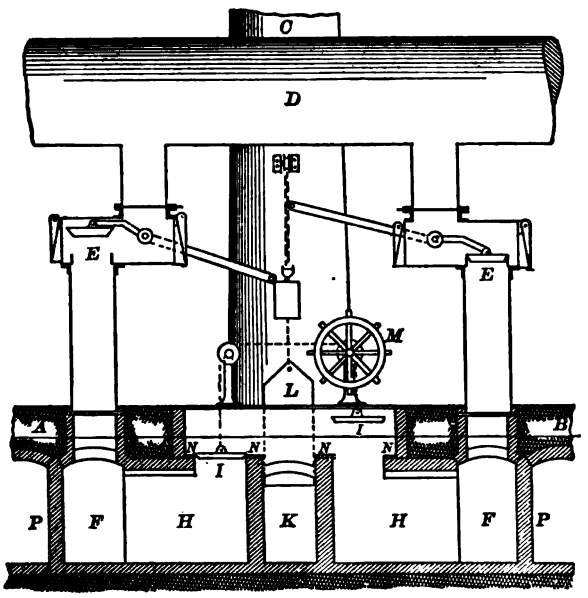


FIG. 238. REVERSING VALVES FOR SIEMENS REGENERATIVE FURNACE.

Vertical section.

C, stack; D, main gas tube; EE, branch gas tubes, showing valves; FF, gas chambers; HH, gas chamber flues to reversing valves; II, stack reversing valves for gas; K, flue from reversing valves to stack; L, stack damper for gas; M, valve reversing gear; NN, water cooled valve seats; PP, air chambers.

the recuperative chambers, the arrangement and dimensions of the latter, the speed of the gases in passing through them and the length of time between reversals. By giving the chambers a sufficient volume and the hot gases a slight velocity they may be made to issue comparatively cold, while the fresh air and gas may be raised correspondingly to a high degree of temperature. According to Friedrich Siemens,[2] the weight of the brick

[1] Trans. Am. Inst. Min. Eng., XXII, 362and 363.

[2] Collected Works, I, 227 and 228.

filling of each pair of regenerators should be theoretically 16 to 17 times the weight of the coal burned between two reversals in order to take up all the heat of the gases of combustion. Consequently in the combustion of 1000 lb. of coal per 24 hours, or about 42 lb. per hour, there should be $17 \times 42 = 714$ lb. of brick in each pair of regenerators when the gas currents are reversed at intervals of one hour; and about 360 lb. at half hour intervals. In practice, however, the whole checker-work is not heated and cooled uniformly, but by far the larger part of its depth is required to effect the gradual cooling of the products of combustion and only a small portion near the top, perhaps a fourth of the whole mass, is heated uniformly to the full temperature of the flame, the heat of the lower portion decreasing gradually downward nearly to the bottom. Three or four times as much brick work is therefore required than is equal in heat capacity to the products of combustion.

The size of the chambers is commonly calculated according to the superficial area that is exposed. Siemens considered that each pair of chambers should expose 51 sq. m. per 1000 kg. of coal burned per 24 hours, or about 6 sq. ft. per pound per hour.[1] According to Roberts-Austen,[2] in order to insure that the gas shall not escape to the chimney at a temperature higher than 150° C. there should be 7 to 7·5 sq. ft. of brick surface for every pound of coal burned between reversals in direction. The brick should be arranged in the chambers so as to leave as much space free as full, i.e., they should not occupy more than 50% of the volume of the chambers. The arrangement of the chamber should be such as will compel the gas to travel uniformly through all parts of it, preventing any tendency on its part to take the most direct course, short circuiting so to speak, and avoiding dead corners. Siemens considered that the chambers were best arranged vertically, heating from the top downward. For various reasons it is preferable to put the chambers beneath the hearth of the furnace when that can be done conveniently. The velocity of the gas through the checker-work may be 1 to 2 m. per second. In good practice the escaping products of combustion are cooled down to about 300° C.

The great advantage of the Siemens system in respect to economy in fuel is referred to in a subsequent section of this chapter. Its disadvantages are the increased cost of the furnaces, the trouble that is sometimes experienced from obstructions in the checker-work, the combustion of gas in the checker-work by leakages between the gas and air chambers, the maintenance of the reversing valves, and the trouble of periodically making the reversals. All

[1] This referred to good Bohemian brown coal.
[2] Introduction to the Study of Metallurgy, fourth edition, p. 297.

of these factors necessitate the employment of more skilful labor than in the case of the ordinary furnace. Attention is sometimes called to the loss of gas that occurs when the valves are reversed, the volume then filling the gas regenerator being diverted to the chimney. However, this is really an unimportant loss, as F. A. Thum pointed out many years ago.[1] He showed that a Silesian distillation furnace smelting 5000 to 6000 kg. of ore and consuming 5000 kg. of heating coal per 24 hours had regenerators of about $3\times1\times6$ m.$=18$ cu. m., half of which was filled with brick and half open. A chamber would contain therefore 9 cu. m. of gas, besides which there might be 3 cu. m. in the flue leading to the chamber, a total of 12 cu. m., indicating a loss of $12\times24=288$ cu. m. at an hourly change of valves. Reckoning that 1000 kg. of coal affords 3500 cu. m. of gas, it is apparent that the loss in this way is comparatively slight, being in fact only $288\div(3500\times5)=1{\cdot}8\%$ at the most. In reality it is less than that because the gas filling the regenerative chamber is at a more or less elevated temperature, wherefore its volume is expanded. With half hour reversals the loss is of course twice as great as with hourly. The loss of gas which occurs in this manner has been obviated, it is claimed, by a special arrangement of valves recently patented by James D. Swindell.[2]

Counter-Current Systems.—In the counter-current recuperative systems the products of combustion are made to escape through one set of flues, while the air for secondary combustion enters through another set of flues surrounding the first, or the combustion gases may surround the air flues. The combustion products and air travel therefore in opposite directions. Ordinarily only the air is preheated, the gas in such installations being delivered to the furnace with as much of its primary heat as possible, but sometimes the gas is introduced in such a way as to be raised materially in temperature. An example of the latter design is described on p. 470. The flues are commonly constructed of thin fire clay tiling, with rebated or broken joints so as to be perfectly tight. It is essential in a recuperator of this type that air be not permitted to escape into the gas flues, or gas into the air flues, which movement in case of leaks will take place in the direction toward which there is the lower pressure. In some counter-current recuperators clay cylinders similar in dimensions to an ordinary Belgian zinc retort with both ends open are employed and sometimes iron pipes are used, as in the stoves formerly employed in iron smelting and now in copper smelting, which will be described subsequently. Another form of counter-current heat recuperator is that in which the air is made to pass through both

[1] Eng. and Min. Journ., 1879, XXVII, 275 et seq.
[2] United States patent, No. 686,781.

clay cylinders and iron cylinders, the latter being employed in the part of the recuperative chamber nearest the chimney, where the products of combustion are coolest and the cold air is admitted. The fuel economizers which are now employed in connection with steam boilers are counter-current recuperators in which water instead of air is circulated through iron pipes around which the products of combustion pass.

Counter-current recuperators are less efficient than those designed according to the Siemens system inasmuch as the heat must be conducted through partitions which have to be made necessarily of poorly conducting material when exposed to very hot gas, while for mechanical reasons fire clay partitions cannot be made thinner than about 1 in. under the most

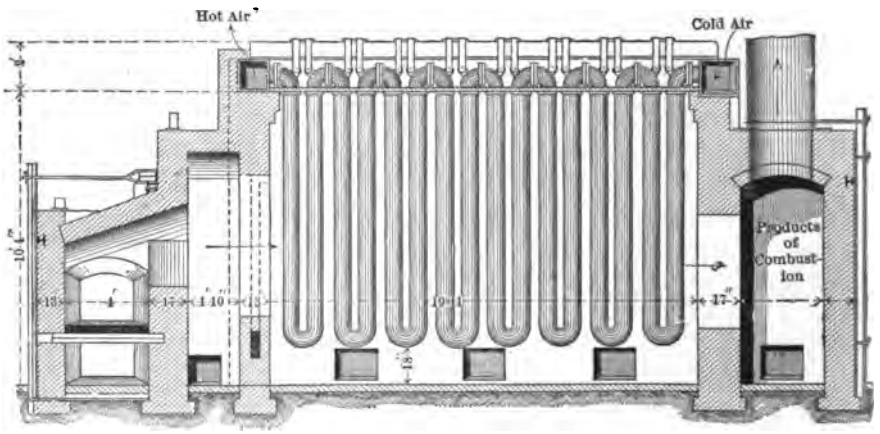


FIG. 239. HOT BLAST STOVE, SIDE VIEW OF U PIPES.

This stove has 64 U pipes, 8 in. diameter inside and 9½ in. outside, suspended from 10 in. I beams.

favorable conditions, and must often be about 2 in. In order to abstract the same quantity of heat as in the direct-contact Siemens system a considerably greater surface must therefore be exposed. The maintenance of the flues is apt to be troublesome; they should be arranged so that there will be easy access to them to make repairs when necessary, but in many furnaces that is not easy to do. For these reasons the counter-current recuperators have not come into so general use as the Siemens system, though there are many types of furnaces, including those for the distillation of zinc, to which they are well adapted and in connection with which they have been employed with great advantage, especially in Rhenish Prussia and Westphalia. An important advantage of this system as compared with the

Siemens is that the direction of the gas currents does not have to be reversed and there is no reversing mechanism to be maintained in repair. The calculation of the proportions for a counter-current heat-recuperative system is too complicated a matter to be entered into here, and any one contemplating such an installation should consult an engineer familiar with the conditions. The same advice may be extended to the design of Siemens furnaces.

U-Pipe Stoves.—The common type of heat recuperator, known as the cast iron U-pipe system, such as is manufactured by the Colorado Iron Works Co., of Denver, Colo., is illustrated in the accompanying engravings. The following data relating to them are supplied by the Colorado Iron Works Co.:

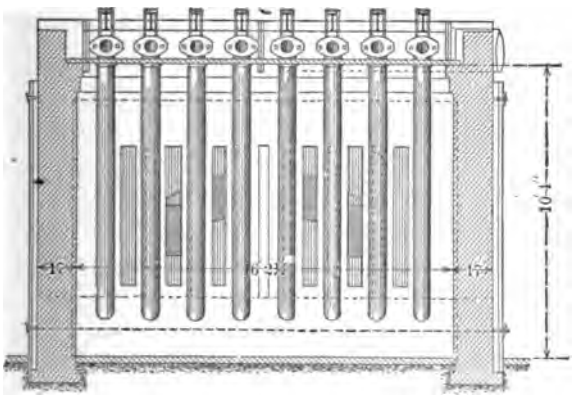


FIG. 240. HOT BLAST STOVE, END VIEW OF U PIPES

Two sizes of U-pipes are used, namely 6 in. and 8 in. in diameter. These pipes are arranged in multiples of 24, as for example, 24, 48, 72 and 96 pipes; or else in multiples of eight as for example 32, 64 and 96. The pipes are 9 ft. long in each leg of the U or 18 ft. in total length, measuring from the flange which supports the tiles of the roof. The pipes are suspended from I beams resting on the end walls of the stove as shown in the engravings. Just below the bends at the top there are flanges cast on each leg of the pipe, upon which fire clay tiling is laid, thereby forming a roof to the flue through which the gases of combustion pass, surrounding the pipes. The elbows and flanges of the U-pipes projecting above this tile roof are usually buried in sand or ashes to prevent radiation of heat from them and are thus easily accessible for repairs or renewal when necessary and more-

over are removed from direct action of the gases of combustion. The joints of the flanges are planed or turned true so that there is no leakage of air.

Another system of uniting the U-pipes is by cemented joints. Small curved U-pipes are made to connect over from one to another of the main U-pipes, which are enlarged at their tops, forming sleeves into which the small curved sections enter loosely. The joints are made tight by cementing with a mixture of iron filings and sal ammoniac. This was the method commonly employed at iron blast furnaces where U-pipe stoves were used before the introduction of the Siemens regenerative stoves, and is by many considered the best way of making the connections. The perfect fitting and strength of the joints are highly important features in a heat recuperator of this type, since leaks greatly destroy the efficiency of the apparatus. All joints must be absolutely rigid, and should have sufficient strength to resist the strain of expansion and distortion of the U-pipes if they should be overheated.

It is very difficult to give trustworthy data as to the life of recuperators of this type, since so much depends upon the conditions under which they are employed. The Colorado Iron Works Co. states that at one place where a stove containing 48 six inch U-pipes was in use the annual expense for renewals did not exceed $125, the breakage being an average of two pipes per annum; at other places the stoves have been run for several years with no breakage of U-pipes and practically no expense for renewals, except new grate bars occasionally. These statements had reference to stoves heated directly from a grate, as shown in the accompanying engravings, which were not therefore recuperators, although by omitting the fireplace they might be used as such.

According to the Colorado Iron Works Co., the heating surface necessary for raising air to a temperature of 600° F. may be taken as 0·4 sq. ft. per cu. ft. of air to be heated per minute; in order to raise the temperature to 800° F. 0·5 sq. ft. should be provided. The external surface exposed by 24 U-pipes of 8 in. inside diameter is 1059 sq. ft., the internal surface being 892 sq. ft. The external exposure of 24 U-pipes 6 in. in diameter inside is 834 sq. ft., the internal surface being 669 sq. ft. The number of pipes must be adjusted not only with respect to the temperature required, but also with respect to the volume of air that is to be transmitted. The Colorado Iron Works Co. considers that the extreme ultimate velocity of the heated air on leaving the stove and in the pipes to the furnace should not exceed 5000 ft. per minute. Since air expands 0·002035 of its volume for each degree Fahrenheit, when heated from 60° F. to 600° F. its volume has become 2·1 times the original volume, wherefore all pipes must have

more than double the area required for the cold air. The Colorado Iron Works Co. has designed an improvement on the ordinary U-pipes, so as to increase their heat radiating surface by casting ribs on the inside of the pipes as shown in the accompanying section through a leg of the pipe. This arrangement increases the heating surface to practically three times that of a plain pipe; cast iron being a good conductor of heat and air a very poor one, the idea is to obtain as much heat radiating surface in a given space as possible. A British invention for accomplishing the same purpose comprises a series of stops with central and peripheral apertures placed alter-

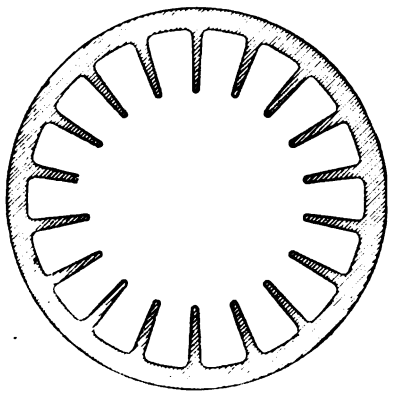


FIG. 241. SECTION OF U PIPE, WITH INSIDE RADIATING RIBS.

nately in the pipe so that the air is continually forced against the heated surface.

The weight and cost of cast iron U-pipe stoves as designed and built by the Colorado Iron Works Co. are given in the following table:

Number of pipes	24	32	48	64	72	96
Weight of cast-iron, 8″.lb...	60,000	80,000	106,000	140,000	167,000	175,000
6″.lb...	40,000	54,000	72,000	96,000	115,000	120,000
Cost of cast-iron, 8″.....	$3,100	$4,300	$5 900	$7,800	$8,500	$10,500
6″.....	$2,100	$2,963	$4,025	$5,300	$6,000	$7,500
Weight of wrought iron, 8″.lb.	7,500	9,500	13,000	16,000	18,000	22,000
6″.lb.	5,000	6,500	9,000	11,000	12,000	15,000
Cost of wrought iron, 8″...	$325	$450	$625	$750	$950	$1,200
6″...	$225	$300	$425	$550	$650	$800
Fire brick required, 8″.....	7,000	8,000	10,000	12,000	14,000	17,000
6″.....	6,000	7,000	9,000	10,500	12,000	15,000
Red brick required, 8″.....	22,000	26,000	29,000	34,000	38,000	44,000
6″.....	20,000	24,000	27,000	31,000	35,000	40,000
Days' labor required to install iron work after brick work is in place, 8″.....	9	12	16	22	26	32
6″.....	7	10	14	20	24	30

Generation of Steam from Waste Heat.—Metallurgical installations are sometimes made wherein the gases of combustion escaping from high temperature furnaces are caused to pass through the flues or around the tubes of a steam boiler. In such cases the boiler is a heat recuperator. Sometimes the hot gases from the furnace are caused to pass first through a stove in which air is preheated, and thence are conducted, at a reduced temperature, to the boiler. It is no new idea to utilize the waste heat of a distillation furnace for the generation of steam, such a scheme having been carried out at the Asturienne works at Auby, France, in the '70's.[1] The chief objection to the practice is the inherent difficulty of carrying out two differently marching processes in combination. When the temperature of the distillation furnaces is allowed to fall off during the maneuver and the gases of combustion escape cooler, it may happen that more steam is required from the boilers, and vice versa less steam may be needed when the gases are hottest. Indeed it would be precisely those conditions which would happen in the natural order of things. It may be laid down as a general principle that since the distillation is the primary process and the generation of steam is only accessory, the management of the boilers must conform absolutely to the requirements of the distillation furnace. Otherwise the increased loss of zinc that may be suffered in distillation may easily offset the value of the steam. This principle implies that if steam is to be generated from the waste heat of the furnace the boilers must have sufficient heating surface to afford the requisite supply at the minimum temperature of the gas; and there must be a system of dampers by which the gas can be diverted from the furnace directly to the chimney when the boilers must be cooled for cleaning or repairs, or into a reserve battery if steam from one set of boilers be required continuously.

Chemical Regeneration.—The latest system of regenerative gas firing invented by Friedrich Siemens was called by him "chemical regeneration"; it was described in a series of patents issued between 1890 and 1899. In this system there are only two regenerative chambers, both for air alone, and the gas producer is directly connected with the furnace so that the heat of primary combustion is preserved. The products of combustion escaping from the furnace are divided. One portion passes through one of the regenerative chambers and thence to the chimney, the air for secondary combustion entering in the meanwhile through the other chamber. The other portion of the combustion gases is continuously blown into the producer, beneath the grate of the latter. In passing through the bed of incandescent coke the carbon dioxide is reduced to monoxide, which is returned to the furnace.

[1] Berg- u. Hüttenm. Ztg., 1880, p. 36.

This cannot possibly create any heat, because as much is absorbed in decomposing the dioxide as is subsequently returned to the furnace, but all of its physical heat, together with that of the nitrogen, save the loss in transit, is recovered, whereas in the ordinary system it escapes to the chimney at a temperature of perhaps 300° C.

Steinmann states[1] that according to information furnished by the London Bureau of Friedrich Siemens there were in 1899 about 300 examples of the new system in the iron, steel and glass works of Great Britain and the Continent; and that according to trustworthy data the economy in fuel is 30% and upward as compared with the ordinary Siemens system, while the first cost of the furnace is 30% less.

Comparative Economy of Direct and Gas Firing.—It is difficult to summarize briefly the economy in the fuel consumption for zinc smelting that has been attained by the introduction of gas firing, because of the insufficiency of the data. The character of the coal employed at one works may be so different from that which is used at another that in the absence of data as to their calorific values and other factors it would be futile to attempt a comparison of results. If a comparison between gas firing and direct firing in the same works be made it is essential to know as to the manner in which each method, but especially the direct firing, is performed. Having in mind that in view of so many uncertainties comparisons from old records are of doubtful value, some idea of the results attained in practice may be obtained from the following paragraphs:

In the case of simple gas firing, the only possible gain to be expected is that which results from the more perfect combustion of the fuel that can be effected and the ability to effect the combustion with a smaller excess of air, besides obviating the admission of unnecessary air during the stoking. The saving that is obtained in those ways is offset to the extent that the heat of primary combustion is lost by radiation from the producer and the flue by which the gas is conveyed from it to the combustion chamber of the furnace. In certain installations, wherein the producer is situated very close to the combustion chamber and radiation is intercepted by the circulation of air around the fire box of the producer, the loss of primary heat is probably very small. The Société Anonyme de la Vieille Montagne experienced at its French works an economy of 10 to 15% in fuel from gas firing as compared with direct firing.[2] The coal burned at those works contains as high as 38% of volatile matter. The introduction of gas firing in connection with Boëtius generators in Rhenish Prussia and Westphalia

[1] Compendium der Gasfeuerung, p. 98.
[2] M. de Sinçay, The Mineral Industry, VIII, 655.

is said to have led to a saving of 30% in fuel.[1] In so far as a general statement can be made, about all that can be safely said is that simple gas firing may lead to a saving of 10 to 30% in fuel as compared with direct firing, depending upon the conditions under which the two methods are applied.

In the case of recuperative gas firing by either the Siemens or countercurrent systems the saving in fuel is commonly greater. The loss of heat in this method of firing need not be theoretically much more than what is radiated from the producer and the gas conduit leading to the furnace and from the furnace itself, plus the heat remaining in the products of combustion to the extent that is necessary to produce the required chimney draught, which is at its maximum when the products of combustion are only 273° C. above the atmospheric temperature. The cooling of the products of combustion is of course a function of the heat absorbing capacity of the recuperative chambers and the nature of the cooling medium (either air and gas, or air alone) which is circulated through them upon reversal of currents, and the efficiency of the system can be varied greatly by the design of the furnace. Similarly the loss of heat by radiation is largely a function of the arrangement of the plant and the design of the furnace. The statements that gas firing by the Siemens system sometimes shows an economy of 50% in fuel as compared with direct firing is by no means unreasonable; it might conceivably be even greater than that. A saving of something like 50% in fuel is said to have been actually experienced in Upper Silesia since the introduction there of Siemens furnaces for zinc smelting. The new Siemens system has not yet been applied to zinc smelting.

MEASUREMENT OF HIGH TEMPERATURES.—The accurate measurement of high temperatures for the control of metallurgical processes is gradually becoming recognized to be a subject of the highest importance. There is probably no branch of metallurgy in which a perfect control of the temperature is of more importance than in the distillation of zinc oxide. Unfortunately the pyrometers which are available for this purpose are too expensive for general application and too delicate for use by any but trained observers. The Le Chatelier thermo-electric pyrometer is probably the best. For a description of it and other types of pyrometers the reader is referred to Roberts-Austen's *Introduction to the Study of Metallurgy.*

The common method of judging the temperature of a furnace is by the color intensity of its incandescence. Providing that the color can be gauged accurately determination in that manner is accurate, since contrary to the general belief there is no difference in the color tints of different sub-

[1] Kerl, Grundriss der Metallhüttenkunde, p. 455 and p. 465.

stances at the same temperature. The eye perceives objects only through differences either in the tint or intensity of light reflected from them. Objects which emit the same tint and intensity of light cannot be distinguished from each other, no matter how different their texture, surface or shape may be. Every object at exactly the same temperature emits the same tint and intensity of light; if not they would remain distinguishable in a furnace when at precisely the same temperature, which is not the case.

The unaided eye, even of a trained and skilful workman, dealing with a special set of conditions with which he is familiar, is a far less trustworthy guide than it is supposed to be, since in estimating the temperature of a furnace, or that of a glowing mass of metal, much will depend upon the relative brightness of the illumination of the surrounding space. The most skilful observer is probably unable to estimate the temperature of a furnace by the eye alone nearer than 50° C. There are several optical pyrometers, including one invented by Professor Le Chatelier and another by Messrs. Mesuré & Noel, which are designed on entirely sound, scientific principles and enable the temperature of incandescent bodies or furnaces to be determined almost exactly on the basis of the intensity of light emitted from them, the personal factor being to a large extent eliminated. These instruments are very convenient and should be more generally employed in the control of metallurgical work than they are.

Color Scale of High Temperatures.—The temperatures corresponding to different colors are now known quite accurately through the investigations of Messrs. Maunsel White and F. W. Taylor, together with those of Professor Henry M. Howe.[1] The results of those investigators have entirely discredited the old and generally accepted table of Pouillet, which is still reproduced in most textbooks and manufacturers' catalogues. The figures of Messrs. White and Taylor and Professor Howe are as follows:

White and Taylor.			Howe.		
Color	°C	°F	Color	°C	°F
Dark red, blood red, low red...	566	1,050	Dull red.....	550	1,022
Dark cherry red.....	635	1,175		625	1,157
Cherry, full red.....	746	1,375	Full cherry.....	700	1,292
Light cherry, bright cherry, light red.....	843	1,550	Light red.....	850	1,562
Orange.....	899	1,650			
Light orange.....	941	1,725	Full yellow.....	950	1,742
Yellow.....	996	1,825		1,000	1,832
Light yellow	1,079	1,975	Light yellow.....	1,050	1,922
White.....	1,205	2,200	White.....	1,150	2,102

[1] Proceedings of the American Society of Mechanical Engineers, December, 1899; Eng. and Min. Journ., December 23, 1899, and January 20, 1900.

DESIGN AND CONSTRUCTION OF FURNACES.—There are certain general principles governing the construction of furnaces in which a high temperature is to be maintained that are applicable to furnaces required for all kinds of metallurgical purposes, whether it be lead smelting, copper smelting or zinc smelting. Some of these principles pertain to the design of the furnace in order to insure an economical combustion of the fuel and the production of the requisite temperature; others pertain to the mechanical construction of the furnace in order to insure that it will have a proper durability with the minimum expenditure for repairs and renewals.

Requirements of a Good Furnace.—A properly designed furnace should have ample coal burning capacity, which means that it should be able to burn the quantity of coal or gas required to generate the maximum quantity of heat that may be required at any stage of the operation, under the most unfavorable conditions that may be expected, such as a supply of coal of poorer quality than is usually furnished and various states of the atmosphere that may tend to diminish the chimney draught if the furnace be worked in that manner. The grate should be of such a kind that ash and clinker may be easily removed from it without stopping the operation of the furnace for more than a few minutes at a time, and the bars should be so spaced that the minimum proportion of coal will be wasted by falling through them unburned. The combustion chamber should be so constructed as to be capable of burning thoroughly all the gases distilled from the fuel before they have a chance to escape to the chimney. It should be therefore of sufficient volume to retain the gases long enough to insure complete combustion and also long enough to enable them to give up all the heat they are capable of imparting. The fire box and combustion chamber should be built with thick walls to prevent loss of heat by radiation so far as possible, care being taken to arrange the construction in such a way that relining can be readily effected.

Both in direct firing and in gas firing it is essential that room be provided for air and gas to mix thoroughly and attain full combustion before the temperature be reduced by the heat abstracting parts of the furnace. This is especially important when the air is supplied cold, and even more so when the fuel is natural gas, which is also difficult to mix with the great volume of air required for its combustion. For this reason, in the earlier distillation furnaces at Iola, Kan., wherein the chief supply of gas was introduced at one end it was customary to leave a space of a few feet between the end wall and the first tier of retorts to serve as a mixing chamber. Both producer gas and natural gas being specifically lighter than air should be introduced into the furnace below the air supply. This is of consider-

able importance in the case of producer gas and air when both are supplied hot, and of great importance in the case of natural gas, which is only about one half as heavy as air, unless the air and gas are to be mixed positively by means of Bunsen burners. In connection with the design of furnaces for producer gas firing reference should be made to pages 296 and 297.

Proportions of Distillation Furnaces.—It is obvious that given the requirements of a certain temperature and a certain quantity of heat and the coal from which those results have to be obtained there should be a certain relation between the grate area of a distillation furnace, the size of the combustion chamber (especially the free space around the retorts) and the draught power which the chimney must have. Gruner laid down the rule that the combustion chamber, or laboratory portion of a zinc distillation furnace, should have 101 cu. ft. capacity for every hundredweight (112 lb.) of coal burned per hour. If then 25 lb. of coal were to be burned per square foot of grate per hour, there should be 4·5 sq. ft. of grate per 101 cu. ft. of combustion chamber, or 1 sq. ft. per 22·5 cu. ft.; the draught must of course be sufficient to effect that rate of combustion. Knab states[1] that direct fired Belgian furnaces should have 1 sq. m. of grate surface to 5 or 6 cu. m. of combustion chamber, or approximately 1 sq. ft. of grate to 16 or 20 cu. ft., while 11 to 12 retorts may be installed per cubic meter (35 cu. ft.). The direct fired Silesian furnaces with 32 muffles, which is the usual number, have a grate area of about 1 sq. m. and a combustion chamber of about 8 cu. m. in volume, of which the muffles occupy scarcely one third. Francis Laur gave the formula that it is not prudent to construct furnaces (meaning single furnaces, not massives) of which the internal capacity exceeds 880 cu. ft.; and that the interior volume of the retorts should be only about one third of the volume of the furnace in which they are heated, experience having demonstrated that approximately two thirds of the interior of the combustion chamber is required for supports and free circulation of the gases.[2]

The ordinary direct fired Belgian furnaces used in Missouri and Kansas until recently, fitted with 224 retorts (112 per side), arranged in seven rows of 16 each, had two combustion chambers, each 17 ft. 6 in. long, 3 ft. 8 in. wide and 9 ft. 9 in. high, the cubic contents being $2 \times 624{\cdot}5 = 1249$ cu. ft. Each combustion chamber was heated by two grates, each 6 ft. $\times$ 16 in., or approximately 8 sq. ft. in area, wherefore the total grate area of the furnace was 32 sq. ft., giving 1 sq. ft. of grate per 39 cu. ft. of furnace. The internal volume of the retorts was 300 cu. ft. and their

[1] Traité de Métallurgie, 452.
[2] Bulletin de la Société de l'Industrie Minerale, 1874, III, 395; Inst. of Civil Engineers, XLI, 317.

external volume inside of the combustion chamber was 554 cu. ft.[1] The interior volume of the retorts was consequently less than 25% of that of the combustion chamber, while the free space of the latter was 1249 − 554=695 cu. ft.=approximately 55·6%. The consumption of slack coal was about 40 lb. per sq. ft. of grate per hour as the average for 24 hours, the total consumption being about 15 tons, but of course this was lower during the maneuver and higher during the period of distillation. The hourly consumption was about 1280 lb.; according to Gruner's principle it should have been 1400 lb.; and no doubt it was fully as much as that during the period of distillation. The number of retorts per 100 cu. ft. was 18; per square foot of grate, seven.

A similar furnace at Collinsville, Ill., fitted with 256 retorts (arranged in eight rows of 16 each per side) had two combustion chambers, each 17 ft. 6 in. × 3 ft. 8 in. × 11 ft.=705 cu. ft., or a total of 1410 cu. ft. There were four grates each 7 ft. 6 in. × 15 in., having a total area of 37·5 sq. ft., or 1 sq. ft. per 37·6 cu. ft. of combustion chamber. The internal volume of the retorts was 342·5 cu. ft.=24% of the combustion chambers. The free space of the latter was 1410—634=776 cu. ft.=55%. The number of retorts per 100 cu. ft. was 18; per square foot of grate 6·8. As in the case of the Kansas furnaces the Collinsville furnace was designed for the use of an inferior grade of slack coal.

The Boetius furnace (semi-gas fired), illustrated on page 449, had two combustion chambers each 23×4×8·2 ft.=754·4 cu. ft., or 1508·8 cu. ft. for the two. Each had two producers of 9·92 sq. ft. grate area, giving 1 sq. ft. of grate per 38 cu. ft. of combustion chamber. The number of retorts was 294, or a little less than 20 per 100 cu. ft., the number per square foot of grate being a little more than seven.

The Rhenish furnace with 216 large retorts, which is described on p. 417, has two Boetius producers, each with 13·2 sq. ft. of grate area. The volume of the combustion chamber of the furnace is 2829 cu. ft., its length being 41 ft. and its cross-section 69 sq. ft. The space occupied by the retorts is approximately 506 cu. ft., leaving 82% of the volume of the combustion chamber for the interior retort supports and for free space. One square foot of grate corresponds to 107 cu. ft. of the combustion chamber. The main flue to the chimney is 14 sq. ft. in section.

The introduction of gas fired distillation furnaces in which the products of combustion traverse the laboratory from end to end has introduced new

[1] There were 224 retorts, 8 in. in diameter inside and 3 ft. 10 in. long inside, thus having an internal capacity of 1·338 cu. ft. per retort. The outside diameter was 11 in. and 3 ft. 9 in. of their length was inside the combustion chamber, or 2·475 cu ft. per retort. 1·338×224=300; and 2·475×224=554.

principles in the design. The distillation furnaces of that type are more or less analogous to the furnaces employed in the manufacture of open-hearth steel. In the case of the latter furnaces it is considered in American practice that the combustion chamber should measure at least 15 ft. in the clear between the ports in order to afford room for the complete combustion of the gas, while in large furnaces a length of 40 ft. is not excessive.[1] These limits refer to furnaces in which the gas and air are mixed at one end, both the gas and air being preheated. The temperature which is attained is considerably higher than in zinc distillation furnaces, rising to 1600° C., while in zinc distillation about 1300° to 1400° C. is all that is required. It would probably be entirely within bounds to construct a distillation furnace 40 ft. long, in which the gas and air are to be admitted only at one end, providing that the free cross-sectional area, i.e., the space unoccupied by the retorts, were sufficiently large to permit the passage of the gases at a moderate velocity. In those furnaces in which gas or air, or both, are admitted at intermediate points that length has been greatly exceeded.

Furnaces Heated by Natural Gas.—In planning a furnace to be heated by natural gas, in which economical combustion is to be effected, it should be borne in mind that in obtaining the same heating effect as compared with producer gas the natural gas must perhaps be supplied with more air,

CAPACITY IN CUBIC FEET PER HOUR OF SMALL ORIFICES AT DIFFERENT PRESSURES.*a*

Pressure.		Diameter of Orifice.					
Inches of water	Ounces	$\frac{1}{32}$ in.	$\frac{1}{16}$ in.	$\frac{1}{32}$ in.	$\frac{1}{8}$ in.	$\frac{3}{16}$ in.	$\frac{1}{4}$ in.
0·80	....	8·0	12·0	15·0	20	30	45
1·00	....	9·0	13·0	17·0	23	34	51
1·20	....	10·0	15·0	18·0	25	36	56
1·36	0·8	10·8	16·0	20·0	27	40	61
1·40	....	11·3	17·0	21·0	28	42	63
1·60	....	11·6	17·5	21·5	29	43	65
1·80	....	12·0	18·0	22·0	30	44	67
2·00	....	12·8	19·0	23·0	32	46	72
2·50	....	13·5	20·4	25·0	34	50	76
2·70	1·6	15·0	21·0	27·0	38	54	86
3·00	1·8	16·4	24·5	31·0	41	62	92
4·10	2·4	18·0	27·5	34·0	46	68	105
5·40	3·2	21·6	32·0	41·0	54	82	122
6·80	4·0	24·0	35·5	46·0	60	92	135
8·10	4·8	26·4	39·5	51·0	66	102	148
9·50	5·6	28·4	42·5	54·0	71	108	160
10·90	6·4	30·0	45·0	57·0	75	114	169
12·20	7·2	31·0	47·0	61·0	78	122	176
13·70	8·0	32·4	48·5	64·0	81	128	182

a From catalogue of Metric Metal Works, Erie, Pa.

[1] H. H. Campbell, Trans. Am. Inst. Min. Eng., XXII, 357.

while on the other hand the volume of the products of combustion will be less.[1] The flues for the admission of air and the escape of the combustion products should be designed therefore in proportion to their volumes. The supply of natural gas to a furnace is governed by the pressure at which it is delivered and the dimensions of the outlet. The table on p. 371 is useful in designing the piping of furnaces for the combustion of natural gas.

The maximum quantity of gas required by the furnace should first be determined and the pressure at which it is to be introduced. The number of gas inlets having been settled upon, arranging them so as to insure a thorough mixture of gas with air, the size of the orifices required can be picked out from the table referred to. The main pipes and principal branches will be of course proportioned to the quantity of gas to be carried. It is inadvisable to draw things too fine in such calculations; it is always better to have the pipes too large rather than too small, since if the former is the case the flow of gas can be checked as desired by valves and there will be a margin to meet special requirements for extra heat in the furnace.

Furnace Masonry.—In the construction of a distillation furnace, or any kind of metallurgical furnace in which a high temperature is to be attained, it should be observed that all portions of the masonry which are exposed to even a comparatively moderate heat must be laid in clay, instead of with lime or cement, common brick clay (tempered with sand) being good enough for most purposes. Fire brick are best laid, however, with a thin mortar consisting of finely ground raw clay and burned clay (chamotte), one half of the former and one half of the latter. The mixture should all pass a 30-mesh screen. The brick are well laid by first dipping into liquid clay mortar or slurry of the above composition, pressing the brick closely together in the walls and grouting the joints with the same mortar as the work proceeds. The joints in any furnace should be as tight as it is possible to make them, each brick being tapped down to its bearing with a mallet or light hammer. This is an advisable precaution inasmuch as the clay joints are more fusible than the bricks themselves. The reason why lime mortar may not be employed in furnace construction is that at a temperature of 812° C. calcium carbonate is decomposed into lime and carbonic dioxide, wherefore the mortar loses its element of strength; also when lime mortar is exposed to the action of gases containing sulphurous anhydride the calcium carbonate is converted into calcium sulphate which crumbles and consequently destroys the efficiency of the mortar. Lime mortar, which is always much improved by the admixture of about 10% of good cement, may be safely employed, however, for the outside work and wherever there

[1] Vide, p. 335 and p. 338.

is no danger of heat; it makes better looking and stronger work than clay mortar and is always preferred by brick masons, whose inclination to use it in places where it ought not to be has to be guarded against in furnace construction.

The walls of a furnace in which only a moderate temperature is to be attained, as for example a roasting furnace, should be made at least 13 in. thick, i.e., 1·5 brick, in order to prevent excessive loss of heat by radiation. In furnaces wherein a higher temperature is to be maintained, as in the case of distillation furnaces, the end walls should be at least 18 in., or two brick, thick and preferably 27 in., or three brick; the front walls in that particular kind of furnace are necessarily very thin and there is unavoidably a great loss of heat by radiation from them. Roof arches are commonly made 9 in. thick. It is sometimes argued that the roof should not be made thicker than that even in high temperature furnaces, in order that radiation may keep the temperature of the arch below the point of softening so as to prevent its destruction, but that danger is much overrated and with proper construction in other respects the roof may be made of any desired thickness. In many furnaces the side walls are carried upward above the arch and the spandrels are filled in so as to make a level top, as appears in Figs. 295 and 296 and others in Chapter X.

The roof arch of a furnace should be designed with a moderately short radius, especially in arches which are to be exposed to very high temperatures. In the case of roasting furnaces, where the temperature is less extreme, arches with a rise of only 12 in. in a chord of 16 ft. (the radius being 32 ft. 6 in.) are quite feasible to construct and are perfectly durable in operation,[1] but nothing of that kind should be attempted in a distillation furnace, the arch of which may well be given a rise of 1·5 in. per ft. of span. In the case of roasting furnaces the arch has to be constructed of fire brick only for a moderate distance beyond the fire bridge, after which common red brick may be used. Both fire brick and red brick should be dipped into liquid clay mortar before laying, and in laying each brick should be pressed closely against its neighbor and settled in position with a few light blows of the hammer. If red brick are being used for building the arch a moderately soft variety is preferable, especially for the construction of large arches, because hard-burned brick, though stronger, are liable to be too much warped and irregular to be used safely; in any case the brick should be all carefully selected beforehand and assorted in such a manner that each longitudinal row, extending the entire length of the furnace, will be composed of brick of about the same thickness. With fire brick these

[1] For moderate widths a rise of 0·5 in. per foot of span is often sufficient.

precautions are unnecessary since they are made more carefully and are generally quite uniform in shape and dimensions. The masonry is always better when it consists of fire brick alone and this may often be done, where fire brick is cheap, without greatly increasing the cost of the furnace by using the inferior grades in parts which are not exposed to high temperatures.

The brick in the arch should be laid as headers with the 4·5 in. side parallel with the longitudinal axis of the furnace. Every alternate course is begun with a split brick, which being only 1¼ in. thick causes every row across the furnace to break joints. An arch laid in this manner is more stable than when the 2·5 in. side of the bricks is laid parallel with the longitudinal axis of the furnace. The preservation of the proper curvature of the arch is insured by the occasional interpolation of a longitudinal row of wedge-shaped or common key brick called "bull-heads." These are indispensable for the center row when the final keying of the arch is effected. The key brick should be driven so tight as to raise the whole roof off the center on which it is built. The skewbacks of the arch should be laid with specially molded brick, which are both cheaper and more durable than standard brick cut to the requisite shape. In furnace construction in general the cutting and chipping of brick should be avoided so far as possible. Many of the odd shapes that are needed are made regularly by fire brick manufacturers under the names of key, split, wedge, soap, checker, jamb, skew, etc., and special shapes are made to order at but little advance over the cost of the standard, weight for weight.

Buckstaves and Tie-Rods.—The brickwork of a metallurgical furnace must be bound securely with iron, generally in the form of beams for buckstaves and round iron for tie-rods. The beams may be either the standard form of I beam or sections of T rails; the former are the cheaper, affording equal strength with much less weight. The dimensions of the I beams used for buckstaves will depend upon the force which they have to resist, i.e., the expansion of the furnace, the length between supports, i.e., between tie-rods, and their distance apart. In ironing an ordinary single hearth reverberatory roasting furnace 5 in. I beams are usually strong enough, one being set on each side of every working door; distillation furnaces and multiple hearth roasting furnaces will generally require heavier beams, however. The dimensions of individual buckstaves depends of course upon their length and number, i.e., the spacing apart. Cast iron buckstaves are sometimes used, especially in the construction of distillation furnaces in Kansas (where they serve a double purpose), but generally speaking steel is preferable. It will bend if excess of strain come upon it, whereas the rigid

cast iron will break. Cast iron buckstaves must therefore be provided much stonger than in the case of steel.

Lower tie-rods are made commonly of 1 in. round iron. They are frequently simply hooked into the foundation of the furnace, but in cases where it is practicable it is preferable to put a single rod, with a loop at each end, clear through the furnace. If they can be passed through open channels provided in the brickwork, as can always be done easily in the case of long reverberatory furnaces, that makes the best arrangement. The precaution must always be observed not to imbed rods in the brickwork too near the fire, to avoid the danger of burning them. In distillation furnaces it is commonly the case that tie-rods cannot be passed through the furnace, or even into the lower walls. The lower ends of the buckstaves are then held by heavy flagstones, or a horizontal rail or beam, firmly imbedded in the ground and against them. In adopting this expedient there is no trouble from expansion of lower tie-rods, because there are no lower tie-rods.[1] Upper tie-rods have to be 1¼ in. in diameter even for moderate lengths, unless there be a good many of them, and where they have to be threaded to make a joint the ends should be upset so that the thread will cause no diminution in strength. The upper tie-rods may be looped over the ends of an opposite pair of buckstaves and united by means of a turnbuckle, or they may be hooked into U-shaped yokes looped over the buckstaves which can then be tightened by turning up a pair of nuts. Either of those methods is good and greatly superior to simply looping the rods over the buckstaves and tightening them by means of wedges, which are slovenly and likely to slip, or to boring a hole through the end of a buckstave for the rod to pass and be tightened by means of a nut and washer.[2] However, if the buckstaves are of cast iron they may neatly and strongly be made with eyes for the rods to pass through. In order to provide the necessary resistance it is frequently advisable to arrange the buckstaves in pairs. In that case the tie-rod will pass between the two buckstaves and through a horizontal bar behind them, being tightened by a nut screwed up to the latter. Sometimes each pair of buckstaves will be connected by two rods, in which case the buckstaves come between them and the arrangement is quite simple. In any case, the buckstaves should be sufficiently long to permit the upper tie-rods to clear the roof of the furnace, when cold, by several inches, inas-

[1] Geo. A. Davis recommends wedging the foot of the buckstave in an excavation 18×18×18 in. below the floor line and filling up with concrete (Handbook of Chemical Engineering, II, 91).

[2] The distillation furnaces in Kansas are seldom braced longitudinally above the roof, the rods being carried along the fronts of the furnace, resting on lugs cast with the buckstaves, or standards. In the later constructions at Iola they connect horizontal beams at the ends of the furnace. The horizontal beams press against vertical bars set in the brickwork.

much as expansion will cause the roof to rise, when hot, and the tie-rods should never rest upon it.

It occasionally happens in furnace construction that pairs of buckstaves which are necessary in certain places, as for example at the ends of solid walls, cannot be connected directly because of a chimney intervening, or some other obstacle which cannot be surmounted by the upper tie-rod. The draughtsman will take pains to space the buckstaves so as to avoid such interferences, but if it be impossible to do so, the expedient is to put a horizontal beam of suitable size behind the buckstaves and loop the tie-rods around it in such lines as are clear.

The stress of the arches should be distributed by cast iron beams, or steel rails, channel or angle beams, of suitable size bearing against the buckstaves and directly behind the skewbacks of the arches. The distillation furnaces of Kansas commonly have cast iron buckstaves (which serve a double purpose) spaced 26 in. apart and tied across at their tops by ¾ or ⅞ in. rods. A 4×4×¼ in. angle beam distributes the thrust of the arch. In other furnaces the buckstaves are further apart and a heavier beam, sometimes a T rail, backs up the arch.[1]

In roasting furnace construction, and other furnaces of that character, the plan is sometimes adopted of encasing the furnace on all sides with cast iron plates. This increases the strength and durability and presents a neater appearance. There is the disadvantage, however, that the iron casing radiates more heat than the naked brick walls do. In the case of cylindrical furnaces, like gas producers, lime kilns, etc., the structural advantages of building the shaft inside a circular envelop of iron or steel are considered to outweigh the disadvantage of increased loss of heat by radiation.

Whatever be the method of ironing the furnace, the rule should be to put the buckstaves, or the enclosing plates, first in place and build the masonry against them. Beams backing the skewbacks of arches, and those set elsewhere to distribute the thrust of the masonry, should be put in position when the walls attain the proper height. In this way the brick can be set firmly against the iron work and uniformity of bearing is secured. It should not be attempted to fit the external ironwork to the masonry after the latter has been erected.

The expansive force to which a high temperature furnace is subjected is very great. When any solid body expands by heating, the force exerted is equivalent to that which is required to expand the same substance to the same extent by mechanical means. According to Trautwine, each 12 to 15° of heat on the Fahrenheit scale (equivalent to 6⅔ to 8⅓° C.) produces an

[1] Vide drawing of Bertha furnace on p. 442.

expansion in wrought iron corresponding to that produced by a tension of 2000 lb. per sq. in. of section. For each 100° C. wrought iron expands 0·001166 of its length and cast iron 0·001001. Trautwine gives the lineal expansion of fire brick as 0·00049 per 100° C., while Kent states that brick masonry expands from 0·00046 to 0·00089 of its length in the same range of temperature. Assuming a ratio of 0·0005, a furnace 80 ft. in length, heated to 1300° C. would expand $13 \times 80 \times 0{\cdot}0005 = 0{\cdot}57$ ft., or nearly 7 in.

A mathematical determination of the stresses due to expansion in a metallurgical furnace would be rather a complicated problem. The hearth and certain of the walls are solid bodies, but other portions of the furnace are open, permitting free expansion interiorally. Other portions have an opportunity to bend; thus the arch, if properly restrained from pushing out, will rise. Expansion in the furnace is, however, a real force and the buckstaves may be considered as beams, supported at both ends and more or less eccentrically loaded.

In furnaces of ordinary dimensions, properly ironed, the expansion of the brick work is met by releasing the tension of the rods a little when the furnace is first heated up, increasing it again when the fire is let out, and if this is properly attended to the shape of the furnace will be well preserved even after a long campaign. With very long furnaces, however, such as have been built for roasting mechanically and for distillation at Iola and elsewhere, the expansion of the brickwork is an important consideration. If not properly arranged for, the furnace is apt to be badly thrust out at the ends soon after it has been heated up. Some striking results in that way were to be observed in the early experience at Iola, Kan. In some of the later furnaces built there spaces were left in the middle longitudinal wall and in the roof arch to permit the expansion to take up. For a similar purpose the arches of the long mechanical roasting furnaces are laid with checker-work openings in certain courses at intervals in the length. A device adopted by the manufacturers of the Ropp furnace is referred to on page 107. In some high temperature furnaces, e.g., open hearth steel furnaces, the ends of the buckstaves are not held rigidly by the tie-rods, but are backed up by powerful spiral springs, which yield somewhat under the expansive force.

Iron work in the interior of a furnace, such as grate bars, grate bearing bars, etc., should be set with the ends sufficiently free to permit of their expansion, which is twice as great as that of brick; local and unnecessary thrusts in the masonry will thus be avoided.

The method of ironing a reverberatory roasting furnace has been so clearly described by Doctor Peters in his treatise on *Modern Copper Smelting* that

it is useful to repeat his words, inasmuch as the directions which he gives for that process in connection with the construction of a reverberatory roasting furnace apply in many respects to the construction of other kinds of metallurgical furnaces. The lower tie-rods consisting of 1 in. round iron, looped at one end and hooked or otherwise tied at the other are built into the walls when the latter are laid and when the furnace is completed the loops are protruding from the walls. The time for ironing the furnace having come, the buckstaves cut to the proper length are slipped through the loops and temporarily wedged therein to keep them perpendicular.

"The upper tie-rods may be made the same as the lower, with a loop at each end—the necessary tightening being effected by flat iron wedges; or they may have a threaded extremity at one end passing through a corresponding hole in the buckstaff, and fitted with a strong nut; or, best of all, a small ring is formed at one end of the tie-rod, through which slips a U-shaped piece of round iron, which fits against the buckstaff, on the other side of which a piece of flat iron, pierced with two holes for the free ends of the U is held, these ends being threaded; a nut for each of the ends completes the apparatus, and presses the piece of flat iron tightly against the upright. This is a simple and highly satisfactory device, and avoids the disagreeable process of wedging in the one case, or of punching a large hole through a narrow rail in the other. The strain is distributed over two bolts and nuts, and can be instantaneously increased or diminished; nor will the nuts rust solid into place, provided they are saturated with oil annually, and slightly turned, to free them.

"Whatever method of tightening the tie-rods may be selected, the process of ironing or anchoring should begin with the first tie-rod on the *main body* of the furnace, nearest the fire end, and proceed systematically toward the rear, thence returning to the shorter transverse rods that support the arch over the grate, and terminating with the long longitudinal rods, which, for convenience of handling, should be in three lengths, connected with hooks and eyes. Up to this time, no great strain should be put upon the rods, everything being merely brought to a solid bearing; but after all are in place, and the buckstaves evened both vertically and laterally, the rods may be drawn to the desired tension, the skewback being still further supported by a bar of 1×4 in. flat iron, or better, an iron or steel rail, let in flush with the brick work.

"This is largely a matter of experience, and being of vital importance should receive the most careful attention on the part of the builder, as too lax a condition of the rods may permit the entire falling in of the arch,

while the contrary fault may cause a positive buckling and elevation of the same, accompanied with a general cracking and distortion of the lateral walls. The latter accident, in a moderate degree, is much more likely to occur than the former, owing to the natural tendency to overdo a measure essential to safety and yet not exactly defined.

"The lateral rods should be tightened until they begin, when struck near the center with a hammer, to vibrate rapidly, and to be but little depressed when stepped upon. (It is almost needless to say that none of the upper rods should touch the arch.) A simultaneous examination of the brickwork forming the upper portion of the side walls should also be made, as it is there that the effect of the curving of the buckstaff from too great tension, and consequent pressure against the mason work, is first visible. The extreme limit of tension is reached when the first signs of this appear, as nothing can be gained by bending the uprights, and if the latter are sufficiently strong and numerous the arch may be considered perfectly supported. All the rods should be tightened to about the same extent, although it must be remembered that the great length of the longitudinal rods may prove deceptive in estimating their tension, it being impossible to tighten them to such a degree as the shorter lateral ones.

"A single additional precaution is recommended, though seldom practiced by builders. This consists in breaking up a few thin roofing slates into fragments a couple of inches in length, and driving these with moderate force into whatever crevices may still be found in the surface of the arch. Some 20 or 30 pails of liquid mud are now poured over the arch, and the process repeated as it dries until every crack and crevice is filled, and the roof rendered completely solid and air-tight.

"The wooden center on which the arch was built should now be removed by first knocking away the little posts that support it, using a light stick of timber as a battering-ram, and proceeding from one side door to the next until every stick and batten are removed. They should be stored for future use. Any indications of settling on the part of the arch must be immediately counteracted by tightening the tie-rods; but when the precautions enumerated above have been carefully observed this can never occur.

"The length of time the completed furnace may now stand untouched with advantage to the mason work is only limited by the requirements of the business, which almost invariably demand its being put in commission at the earliest possible moment. Under such circumstances a smoldering fire of large logs, knots, or any slow-burning waste material, should first be kindled on the floor of the ash pit, the grate bars not being put in place until the masonry surrounding the fireplace is partially dried. In 12 or 18

hours the fire is elevated to its proper place, and with a nearly closed ash pit door and partially lowered damper, the process of drying proceeds gently and without that violent generation of steam and vapor that is sure to be accompanied by extensive fissuring of the brickwork and permanent weakening of the entire structure.

"A careful examination of the condition of tie-rods and buckstaves should be made every few hours from the first kindling of the fire until the furnace has attained its full heat and may be supposed to have expanded to its utmost limits, although it may be a month or more before all evidences of movement cease. The first indication of this process will be seen in the neighborhood of the bridge and fireplace, where the highest temperature prevails. A bending of the buckstaves, combined with a pressing in of the skewback line and an increased tension of the cross-rods, are warnings that may soon be followed by either a complete giving way of some portion of the ironwork, or more frequently by a bodily upheaval of the arch and general fissuring of the brickwork unless relieved by diminishing the strain to a corresponding degree. This process of loosening must be extended to the entire ironwork of the furnace, and continued as long as necessary, the tension being again increased if the furnace is ever allowed to cool down to any considerable degree—an operation more destructive to it than many months of ordinary wear."

X.

DISTILLATION FURNACES.

The zinc distillation furnace considered apart from the gas producer, or fireplace, and the heat recuperative system and chimney, is simply a combustion chamber in which the fire gases are burned around the retorts containing the ore, the retorts resting on shelves inside the chamber. The furnace is therefore simply a reverberatory heating furnace in which the charge is contained in closed vessels and does not come into direct contact with the flame. All types of zinc distillation furnaces are included under this classification; the various types differ essentially only in the arrangement of their retorts and in details of construction. At the present time it is indeed difficult to draw any sharp distinction between the distillation furnaces which are employed in various parts of Europe and America.

CLASSIFICATION OF FURNACES.—Distillation furnaces were formerly classified as: 1, English; 2, Carinthian; 3, Silesian; 4, Belgian-Silesian; and 5, Belgian. The English and Carinthian furnaces were quite distinct types, the distillation vessels used with the former being large pots from which the vapor passed downward, while in the Carinthian furnaces vertical tubes were employed. Neither of these furnaces has been used, however, for many years. The descriptions of the English furnace which are commonly to be found in the textbooks and are reproduced briefly in this treatise, because of their historical value, are due to Doctor Percy, who saw one of them in operation in 1859; even then they were considered rare. The Carinthian furnace disappeared even a longer time ago; indeed so long ago that no descriptions of it are now to be found in the general metallurgical treatises, but about 20 years ago a modified Carinthian furnace was introduced by Binon and Grandfils, which did not, however, come into general use, and their idea has long been abandoned. The Silesian furnace proper has also gone out of use, having been displaced by a form first introduced in Belgium, where certain modifications from the old Liège type were grafted on the original Silesian furnace. The later form of Silesian furnace is referred to in some treatises as the Belgian-Silesian,

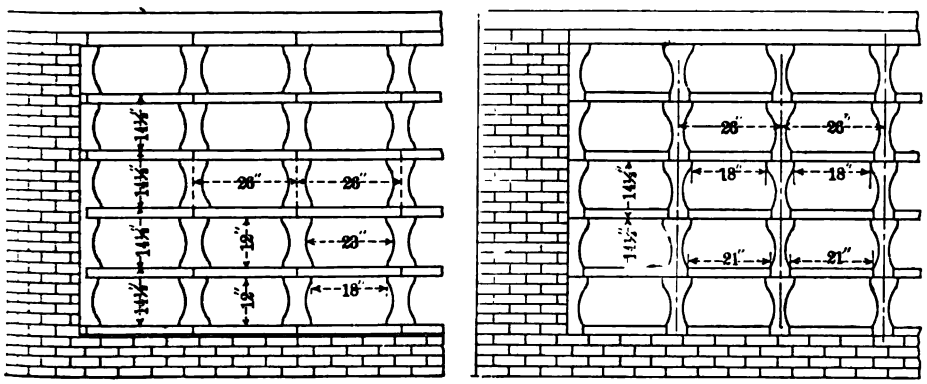
which is confusing inasmuch as the furnace commonly used in Rhenish Prussia and Westphalia, which stands midway between the Belgian and Silesian types, is also referred to as a Belgian-Silesian furnace.

In this treatise the types of distillation furnaces in use at the present time are classified as: (I) Silesian; (II) Rhenish; and (III) Belgian; said classification being adopted merely for convenience, since there is no radical distinction between the Silesian and Rhenish furnaces and the Rhenish and Belgian. Each of these types is subdivided into three classes: (a) direct fired; (b) semi-gas fired; (c) gas fired. The gas fired and semi-gas fired furnaces are further subdivided into: (1) those in which heat is recuperated, and (2) those in which heat is not recuperated. In making the prime classification of Silesian, Rhenish and Belgian furnaces, the only possible distinction is with respect to the retorts used with them and the manner of their arrangement. In each type the retorts are placed horizontally, or rather with a slight slope toward the front. The typical Silesian furnace has one row of large muffle-shaped retorts. The typical Rhenish furnace of the present day has usually three rows of elliptical or muffle-shaped retorts of moderate size. The Belgian furnace has commonly four or more rows of retorts of circular or elliptical section. There have been, however, Silesian furnaces constructed with two rows of comparatively small muffles, and such furnaces have been very similar to the Rhenish furnaces; in fact, almost identical with the original Rhenish furnaces. On the other hand, there are Belgian furnaces, especially the Siemens-Belgian, which have retorts of about the same dimensions as are used with the Rhenish furnaces and do not differ essentially from the latter type. There has been in fact a steady drawing together of the original Silesian and Liège types of furnaces, each borrowing from the other, and modern furnaces are likely to become even more similar.

General Features of Construction.—The distillation furnaces of various types which are in use at the present time are essentially long, rectangular chambers of refractory material covered with an arch springing from the long side walls. In the Belgian type of furnace this chamber is commonly divided into two equal parts by a middle longitudinal wall, and sometimes into four parts by a middle transverse wall (the latter arrangement being found frequently in Belgium, but rarely in the United States), and in those cases the entire furnace is generally referred to as a "block" or "massive," the two or four divisions, as the case may be, often being called "furnaces." This frequently leads to confusion in referring to the capacity of a works. It is better to consider the entire block or massive as the furnace unit, since although each division may have its own operat-

ing crew, the block constitutes a single structure and no one of its divisions would likely be permitted to go out of use while the others were under fire.

In the case of the Belgian type, the furnaces were built originally with only one brigade of retorts; i.e., they had only one working face, and such single furnaces are still to be found in the United States at Pulaski, Va., and at certain works in the West where shelf burners for roasting the ore are built on an arch sprung from the rear walls of two adjacent furnaces facing in opposite directions; also in the case of some of the natural gas fired furnaces in Indiana and in a few instances at Iola, Kan. In Belgium, too, examples of this type still exist. Single furnaces are less advantageous than double furnaces, because their first cost is proportionately more, while



FIGS. 242 AND 243. METHODS OF SETTING FRONT LINING BLOCKS IN BELGIAN FURNACES.

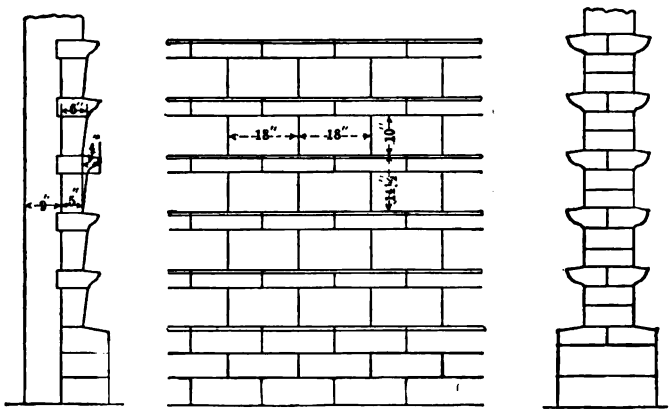
In Fig. 242 the pillars and shelves are bonded together. In Fig. 243 the pillars form a continuous column and the shelves rest on their projections.

they occupy a relatively greater floor space and expose an additional side for the radiation of heat. The modern furnace of all types, Silesian, Rhenish and Belgian, has two brigades of retorts, facing in opposite directions and making each long side of the furnace a working front.

The Brick Work.—The brick work of a distillation furnace consists of two parts, the shell and the lining, these being respectively the outwalls and the inwalls. The shell may be built of common red brick, but the lining must necessarily consist of fire brick of a superior quality in order to withstand the high temperature which has to be maintained inside of the furnace. Practically the only important part of the combustion chamber which can be built of red brick is the outer shell of the end walls. The

latter are built substantially, often with a thickness of 25 to 30 in., and seldom less than 21 in.

The side walls (or front walls) of a furnace are built necessarily in the form of window frames or pigeon-holes for insertion of the retorts, the horizontal members of these pigeon-holes forming shelves on which the front ends of the retorts rest. The pigeon-holes are built up of fire clay blocks and tile, which must be arranged so that they can be removed for relining the furnace without disturbing the arch which constitutes the roof of the furnace. In the original Liège furnaces, which were very short, an arch was sprung from end wall to end wall, and some old furnaces showing that form of construction are still to be found in the United States at the works at Carondelet (St. Louis), Mo. The design was improved,

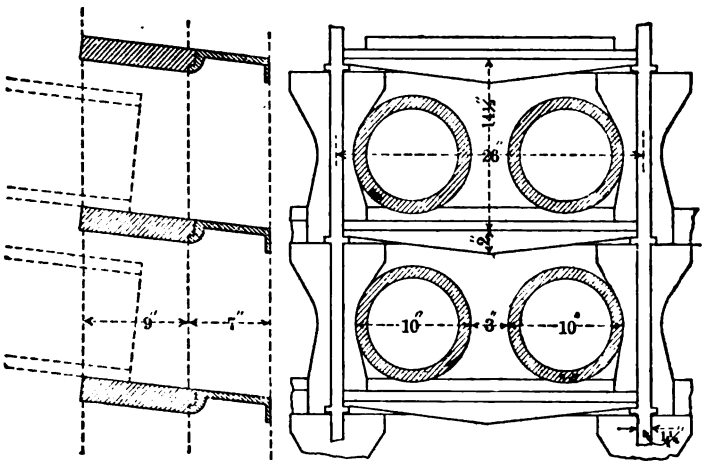


FIGS. 244 TO 246. METHOD OF SETTING REAR LINING BLOCKS IN BELGIAN FURNACES.

Fig. 244: Side elevation, ordinary method. Fig. 245: Front elevation. Fig. 246: Side elevation, bad method.

however, many years ago by setting cast iron stanchions against each pillar of the fire brick lining, and supporting the skewback of the arch on them. Each stanchion was cast with flanges corresponding to the front shelves of the fire clay lining and cast iron plates were placed upon them, thus forming a continuation of the fire clay shelf and giving sufficient width to support the condensers. The façade of the furnace thus presents the appearance of a series of cast iron pigeon-holes superimposed upon the fire brick pigeon-holes. The furnace is stiffened by buckstaves outside the pigeon-holes, which are securely anchored in the foundation at their lower ends and tied across the furnace at the upper ends. The end walls may be supported by buckstaves and tie rods in the usual manner; or there may be

no vertical buckstaves against the end walls, support being furnished the latter by horizontal rods or beams which are tied together at the ends by longitudinal rods resting on lugs cast with the stanchions (or buckstaves) of the furnace front—a form of construction which is exemplified in most of the direct fired Belgian furnaces, and the natural gas furnaces, of Kansas. It is a common construction, especially with the Belgian furnaces used in the Western States, to make the stanchions of the front wall taller than the roof of the furnace, tieing them together at the upper ends so that they act as buckstaves; in this case it is usual to cast the stanchions with brackets protruding inward on which to support an angle-bar carrying the skewback of the arch.



FIGS. 247 AND 248. ARRANGEMENT OF NICHES FOR RETORTS IN BELGIAN FURNACES.

Fig. 247: Vertical section through center. Fig. 248: Front elevation.

The arch is sometimes sprung from side wall to side wall, and sometimes from side wall to middle wall and middle wall to side wall. In the former case the only function of the middle wall is to serve as a support for the inner ends of the retorts. The middle wall is usually built of fire brick, which constitutes the permanent structure, and is enclosed with fire clay blocks of which certain courses protrude into the furnace to form the shelves for the retorts. The middle wall should have a total thickness of as much as 30 in., but in the Iola type of gas fired furnaces it is built generaly 21 in. thick and sometimes as thin as 12 in.. there being in the latter case only a single tier of fire clay blocks. It is almost needless to say that such light construction is unsubstantial and not to be recommended.

In the Silesian and Rhenish furnaces there is no middle wall and the arch is always sprung from one side wall to the other. In the regular type of Silesian furnace, which has only one row of retorts, the latter rest directly on the hearth of the furnace. In the standard type of Rhenish furnace with three rows of retorts there are two longitudinal, interior pigeon-hole frames of refractory tile, corresponding to the outside walls, which form the shelves for the inner ends of the retorts. The external pigeon-holes, or "Capellen," which are considerably deeper than in the Belgian furnace, are commonly built with heavy tiling in the manner shown in the engravings of those types of furnaces (q. v.). The Silesian furnace, having only one row of retorts, is simple in construction. The sides are divided by tiles (Grenzsteine) about 4 in. thick, standing on edge, which are arched over in such a way that they present the appearance of a series of windows. The arch is sprung between the walls thus formed. The outer side of the walls commonly has a slight batter inward, and a buckstave bears against each dividing tile. The arrangement of the Rhenish furnace is similar, but there are, of course, horizontal shelves for the front ends of the retorts to rest upon, and to support the condensers. Holes are arranged in the shelves, through which the residues raked out of the retorts drop into the collecting pockets.

Arrangement of the Retorts.—In all types of furnaces now in use the retorts are set with an inclination toward the front. This is done for two important reasons, viz.: (1) corrosive slags are drained to the cooler portion,[1] where their chemical action is diminished; and (2) it makes it easier for the men in discharging and recharging the upper rows. If the retorts were originally set on a level, any sinking of the inner seat would cause the retorts to have an upward inclination, which would be objectionable for both the reasons given above.

In the Silesian type of furnace, wherein the retorts set directly on the hearth, the latter is commonly given a slope of about 5°, which in a length of 1·7 m. corresponds to a fall of 0·155 m. In the Belgian and Rhenish furnaces the slope is varied somewhat according to the ore that is to be smelted. In Kansas and Missouri, the angle is commonly between 5° and 7°, corresponding to falls of 4 to 6 in. in the usual retort length of 48 in.; a fall of 6° (5 in.) was perhaps the most usual in the old direct fired furnaces. In New Jersey and Pennsylvania, where the slags are more corrosive, more inclination is given to the retorts; in the furnace illustrated further on in this chapter the angle is nearly 11°.

[1] The increase in temperature toward the rear end of the retort is indicated by the lower zinc tenor of the residuum removed from that portion; vide Chapter XII.

In the Belgian and Rhenish furnaces, the retorts may be set so that all will be parallel; or the upper rows may have a steeper inclination than the lower; examples of both arrangements will be found in the numerous engravings that accompany this chapter. In some of the older direct fired furnaces it was often the practice to make the lower rows of retorts shorter than the upper ones, inasmuch as they were exposed to the strongest heat, while at the same time the middle longitudinal wall was made correspondingly thicker. Uniformity of retorts is desirable, as is also the fewest possible shapes of the lining blocks, and in the modern furnaces, which afford a more equable temperature, the retorts are usually made of the same length and all are set parallel.

Longitudinally the retorts have to be spaced with such intervals between them as will permit proper circulation of the flames around their exteriors and will leave room between each pair for the pillars, or dividing plates, of the front wall. In the Silesian and Rhenish furnaces the latter are com-

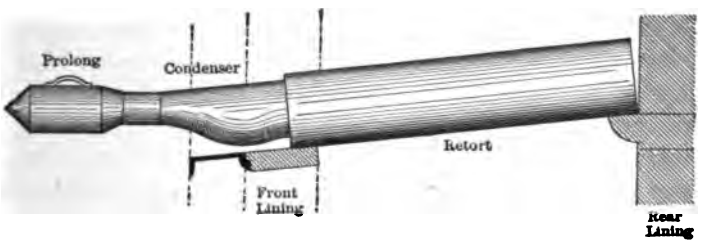


FIG. 249. RETORT, CONDENSER AND PROLONG.

monly about 0·1 m., or 4 in., in thickness and about the same space is left between the two muffles that occupy a niche. In the Belgian furnace the pillars of the front wall are curved to correspond with the retort, and in direct fired, and some gas fired furnaces, they are only 2⅜ to 3 in. thick at the middle. In those gas fired furnaces in which air or gas, or both, are introduced at intervals in the front walls, the pillars through which the ports pass have necessarily to be made wider to allow for them. This leads therefore to more free room in the combustion chamber and increases the length of the latter correspondingly.

The height of the niches has only to be such as will permit easy insertion and removal of the retorts. In the Belgian furnaces of Kansas there is usually a clearance of about 1·5 in. between each retort and the bottom of the shelf next above it. The total height of the furnace depends of course upon the number of rows of retorts. In the older Belgian furnaces there were often as many as eight rows. Nowadays there are seldom more

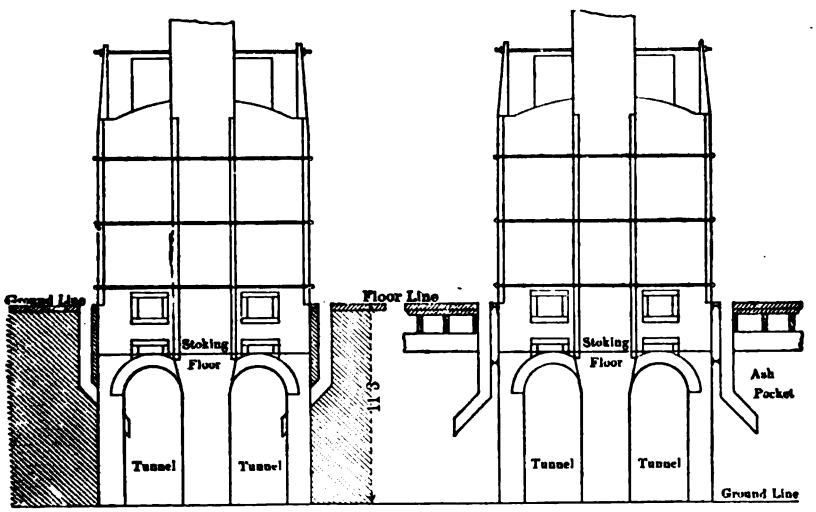
than five, and in Belgium there appears to be a tendency to reduce that number.

The condensers, of which the inner end rests in the mouth of the retort, are supported at the outer end by means of a half brick set on the sill of the niche; or else on an iron bar, supported horizontally by lugs cast with the standards of the furnace front. The Iola furnaces have two small curved castings pivoted in the front shelf of each niche, which when turned into position are just right to hold the condenser as it ought to be; when the condenser is removed in order to discharge the retort, the rest is turned aside and is out of the way. Such a device, of which modifications will suggest themselves, is much neater and more convenient than depending upon a loose and clumsy brick. The Silesian furnaces have a counterweighted door in front of each niche, which can be pulled down or put up as may be required. This device does not find application in the Belgian type of furnace.

The Working Floor and Inferior Structure.—Although the distillation furnace is essentially a simple combustion chamber as above described and furnaces involving no other parts are to be found in use by the natural gas smelters of Indiana, at least, the modern furnace is generally a good deal more complicated structurally, especially if it be equipped with a heat-recuperative system, while if the furnace be of the direct fired type, with a deep fire box, there is necessarily a subterranean structure to give room for the fireplace under the retorts, or else the working floor must be elevated. It is rare to find, therefore, a distillation furnace which consists simply of a combustion chamber built directly on the ground level, the common arrangement with Belgian furnaces being to place the hearth of the combustion chamber, or rather the working floor of the furnace, because in the direct fired furnace there will be no hearth, at an elevation of 9 to 12 ft. above the top of the foundation walls. This gives room for a deep fire box with access to the grate at all parts, and furthermore permits the residues from the retorts to be dropped into hoppers, below the operating floor of the furnace house, from which they can be drawn by gravity into cars. If the furnace be of the recuperative type, it is common to arrange the heat restoring system directly beneath the combustion chamber, and even in the furnaces wherein heat recuperators are built separately, it is customary to build the combustion chamber so high above the ground level that the ashes can be dropped into collecting pockets. If that be not done, the ashes have to be discharged on the floor of the furnace house and subsequently shoveled into wheelbarrows or cars. Such an arrangement saves something in first cost, but involves a good deal of extra labor and incon-

venience in operation and is not to be recommended. The direct fired furnaces which used to be employed in New Jersey were built in that way and the same arrangement is to be observed at most of the works in Indiana.

The modern type of distillation furnace being necessarily so high a structure in order to contain a fireplace and ash pit, or a heat recuperative system, with means in either case for economical handling of the residues, beneath the combustion chamber, it is no simple matter in designing a plant to decide what position the operating floor of the furnace houses shall have with reference to the ground level, especially when the latter is horizontal. The furnace may be made to coincide with the ground level, its lower part



FIGS. 250 AND 251. SKETCH SHOWING DIFFERENT ARRANGEMENTS OF DISTILLATION FURNACES WITH RESPECT TO THE GROUND LEVEL.

being imbedded in the ground; or the base of the furnace may be made to coincide with the ground level and the retorts operated from a platform or an upper floor of the building; or the lower part of the furnace may be built partly under ground and partly above. The first two arrangements are illustrated in the accompanying sketch.

If the entire lower part of the furnace be put under ground, there will be required a more or less extensive system of subterranean galleries with poor light, poor ventilation, and probably cramped room, while the first cost will be considerable; unless the contour of the ground be such that an adit can lead directly to daylight, it will be inconvenient to remove the

ashes from the lower level, and trouble may be experienced from water unless a proper drainage system is provided. On the other hand, if the base of the furnace be made to coincide with the ground level, there is the expense of the elevated floor, and the ore and coal coming from the storage bins must be elevated in some way. The latter is a simple matter, however, since if the ore and coal be mixed mechanically in a separate department, which is by all means the best practice, the charge will have to be elevated anyway, and it is of no consequence whether the height to which it is raised be 10 ft. or 20 ft. The cost of the floor will vary materially, according to the load it is calculated for and the style of construction. It may be more expensive under some circumstances than a system of subterranean galleries, but even if it is the difference will be most likely so small as to have no weight in view of the advantages that are gained. These are a level and permanent working floor, plenty of room around the lower part of the furnaces, good light and ventilation, and no trouble from surface water, while a large additional area beneath the working floor, that may be useful for many purposes, is gained. When the furnaces have recuperative chambers, which are almost always placed beneath the combustion chambers, it is especially desirable to have the whole structure above ground.

In Kansas and Missouri it is the common practice to put part of the base of the furnace under ground and fill in around it up to the operating floor, whereby the furnaces appear to stand on a mound. This is the cheapest arrangement, and as compared with subterranean galleries affords better ventilation, but the ashes have still to be hauled up hill out of the tunnel and the latter is subject to flooding in wet weather. The construction of the furnace entirely above ground is not, however, unknown in the West, since an example of this arrangement is to be seen at Collinsville, Ill. In Belgium, where the distillation furnaces are commonly arranged in line, end to end, in long houses, which are in many cases built on sloping ground, it is a general practice to make the working floor even with the ground level on the up-hill side, and have the sub-galleries lead out on the down-hill side; instead of the galleries, an open room around the furnaces is provided under the entire working floor in some cases. The latter arrangement appears to be the general tendency in the design of the newer works, and where the ground does not slope the working floor is set at the required height above it, being supported by steel beams and brick arches.

ENGLISH FURNACES.—The distillation furnaces formerly used in Great Britain were very similar in design to the pot furnaces which until lately have been employed for the manufacture of glass. The original Silesian furnace was of an analogous type, which was due possibly to the fact that

zinc distillation was introduced into Upper Silesia from Great Britain. Doctor Isaac Lawson is reported to have first invented a practical method of extracting zinc from calamine in Great Britain and to have erected works for that purpose, while according to Pryce zinc works were first erected at Bristol by John Champion, to whom a patent appears to have been granted in 1739, which long antedates the work of Dillinger in Carinthia, Ruhberg in Upper Silesia, and Dony in Belgium. The English furnaces differed from other types by distilling the zinc *per descensum* instead of *per ascensum*.

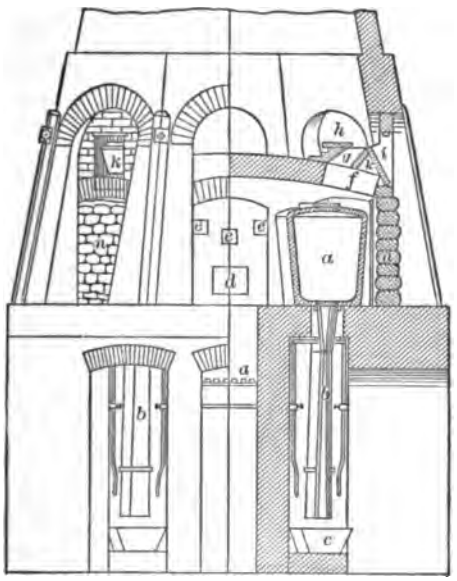


FIG. 252. ENGLISH DISTILLATION FURNACE.

The English distillation furnace was either square or circular in plan, the latter form being preferred. In the center of the hearth there was a fireplace, around which were arranged from six to eight fire clay crucibles, of which the dimensions were: 1·37 m. height, 0·66 m. diameter at the top and 0·36 m. at the bottom, each of these crucibles containing 167 kg. of roasted blende. The bottom of each pot had a hole corresponding to a hole in the hearth of the furnace; the hole in the pot was stopped with a plug of wood, which being converted into charcoal during the process, was rendered sufficiently porous to permit the passage of zinc vapor while preventing the escape of the small coal or calcined mineral. Connecting with

the pot there was a sheet iron tube *b*, which was supported by the arrangement shown in the accompanying engraving, in which *d* is an iron yoke, and *ee* are movable iron rods which are held in position by the set-screws *ff*. A close contact was thus made with the bottom of the pot. After the zinc began to distill off, a longer tube *c* was attached to the short tube *b*. The zinc dripping down from this tube was collected in an iron vessel placed conveniently for its reception. As the tubes were liable to become choked by the condensed metal, it was necessary to clear them from time to time by the insertion of an iron rod, since they might otherwise become entirely closed and give rise to explosions.

The arch of the furnace was 2·13 m. high above the hearth. The flames from the fireplace played around the crucibles and escaped through the

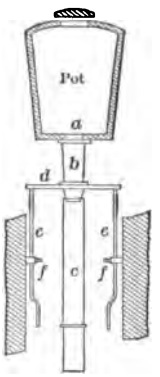


FIG. 253. POT AND CONDENSER OF ENGLISH FURNACE.

flue *g* into the chimney. The crucibles were inserted in the combustion chamber through the openings *n* in the sides, these openings being bricked up after the crucible was in place save for the small holes *ee*, which could be opened to admit air to the combustion chamber or permit repairs to be made on the crucibles. In the arch there were holes over each crucible, through which the charge was shoveled into the crucibles and the covers of the latter were luted on, these openings being closed by the plate *i*. In the engraving, *d* is the stoking door and *h* a damper for regulating the draught.

In operation the crucibles were filled with the proper charge, the covers luted on and the doors *i* in the wall of the furnace closed. At the end of the operation the crucibles were discharged by removing the condensing pipe from the bottom, breaking out with a bar the charcoal by which the

hole was partly closed during the distillation, and discharging the residue through the aperture.

The distillation time ranged from 65 to 68 hours, during which 1016 kg. of blende would yield 304·8 to 406·4 kg. of zinc, with a consumption of 22 to 27 parts of coal per part of zinc. The duration of the pots was calculated to be about four months. The zinc produced had necessarily to be remelted in order to obtain plates of spelter from the solid drop zinc mixed with blue powder and oxide. The high consumption of coal in these furnaces led to their abandonment and none has been in use since about 1859 or 1860.

CARINTHIAN FURNACES.—At Delach, near Greifenburg, in the Drauthale, Carinthia, distillation furnaces with small, vertical retorts were used in the early part of the last century.[1] These were closed at the upper and wider end and terminated at the lower end in a tube through which the zinc vapor passed into a common condensing chamber, the bottom of the latter being formed of an iron plate on which the zinc dropped. The tubes were 1 m. long, 0·114 m. in diameter at the upper end, and 0·082 m. at the lower end. They were charged with 2·5 to 3 kg. of ore per tube, a space 0·1 m. high being left in the upper part of the tube which was filled with small pieces of coal. The furnace, which was heated with wood, comprised 135 retorts, only 84 of them being charged with ore, however. The management of such a large number of small tubes was so difficult and costly, involving a high consumption of fuel and a poor extraction of metal, that this furnace has long been forgotten.[2]

In recent times furnaces with vertical retorts of greater diameter have been proposed and used experimentally, especially for the smelting of zinc ore which contains lead. Binon and Grandfils, Keil, Gruetzner and Koehler and Chenhall have described furnaces of this type.[3]

Binon and Grandfils Furnace.—The furnace of Binon and Grandfils, which is shown in the accompanying engraving, had 12 to 16 elliptical retorts, 2·4 m. long and 0·4 m. in diameter, which were fire clay tubes resting in

[1] Hollunder, Tagebuch einer metallurgischen Reise, Nürnberg, 1824, p. 273; and Knapp's edition of Percy's Metallurgy, p. 550; a detailed account of the practice at Delach may be found in the latter treatise.

[2] The early experiences of zinc smelting in Carinthia are of considerable historical interest, inasmuch as it is not unlikely that the first practical work on the Continent of Europe was done there. Anyway, the industry was established at about the same time as in Silesia. Bergrath Dillinger, of Klagenfurt, who was later appointed director of all the zinc works of Austria, built the smeltery at Döllach, near Grosskirchheim, in the Möllthale, in 1799; in 1801 he built the plant at Delach, the furnaces of which were described by Hollunder.

[3] Binon and Grandfils, Dingler's Polytech. Journ., CCXXXV, 222; Oest. Zts., 1881, p. 325; Rev. univers. des mines, 1879, V, I, 228; Berg- u. Hüttenm. Ztg., LXXX, 10; also Keil, Berg- u. Huttenm. Ztg., LXXXVIII, 116; Gruetzner und Koehler, German patent, No. 58,026, Sept. 25, 1889; Chenhall, Oest. Zts., 1880, p. 462.

the cast iron groove of a box or boot *c*, which was supported on a brick column *b*. The joint between the retort proper and the cast iron boot was made tight by packing with clay. The end of the boot was closed with a plate *d*, which was removable to discharge the residues. In the plate there was a tap-hole for drawing off the lead. The upper end of the retort was closed by the plate *e*, and connecting with the retort near the top was the condenser *f*, supported on the tiles *h*, which formed niches *g*, in the side walls of the furnace. The furnace was heated from a producer, whence the gas was drawn through the combustion chamber in which the retorts stood, the latter being set with their major axes parallel with the long

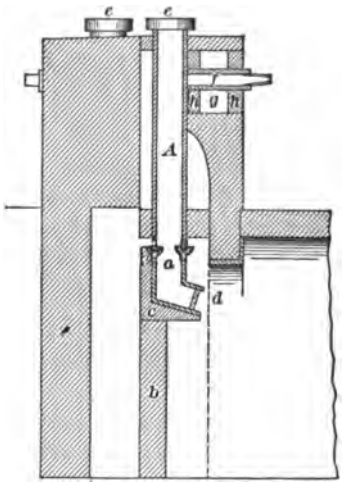


FIG. 254. BINON AND GRANDFILS FURNACE.

axis of the furnace. In operation the lower parts of the retorts, up to the level of the hearth of the combustion chamber, were filled with small coke and exhausted residues; the charge was then put in and the covers luted on. After the distillation was completed, the residues were discharged through the opening *d*, lead having been tapped off in the meanwhile from the tap-hole in the latter and zinc from the condenser *f*. Metallurgical literature contains no record of the economical results obtained from these furnaces, which if my recollection be correct, were tried experimentally at Stolberg, Rhenish Prussia, about 1880. The fact that they did not come into regular use leads to the inference that they were not successful, or at least presented no advantage over the regular furnace of that time.

Messrs. Binon and Grandfils, of whom the former was engineer of the Rhein-Nassau company, considered that the advantages of their furnace would be ability to smelt more corrosive mixtures because of the increased durability of the vertical retorts, saving of time in charging and discharging, and the production of a purer spelter, inasmuch as the reduced lead would sink to the cool part of the retort, where it would not suffer volatilization. Modern practice has demonstrated the commercial feasibility of smelting corrosive ores in the ordinary type of Belgian furnace without danger of producing a spelter excessively contaminated with lead, and of late but little attention has been given to the use of vertical retorts.

Chenhall's Experiments.—The Chenhall furnace, which was invented about 1880 and is referred to in the technical journals of that period, was never used practically. Mr. James W. Chenhall has informed me, in a private communication, that the Swansea Zinc Ore Co. being then engaged in working some ores with a tenor or 40% Zn and 15 to 20% Pb, he conceived the idea that by distilling off the zinc in a vertical retort (gas fired) the lead reduced during the distillation of the zinc would percolate downward through the charge, without penetrating the sides of the clay retort, and would collect in a suitable receiver at the bottom of the retort, whence from time to time it might be drawn off. Experiments on a small scale demonstrated that by effecting the distillation of the zinc in a vertical retort the lead was preserved in a recoverable condition for a far longer time than when it was attempted with the same vessel placed in a horizontal or inclined position. The laboratory experiments indicated so strongly the probable success of the furnace on a larger scale that money was voted for its erection, but for various reasons the plan was never carried out.

Keil Furnace.—This furnace[1] was described by Doctor B. Kosmann in Berg- u. Hüttenm. Ztg., 1888, p. 116 et seq., but he did not state that it had been given a practical trial. It was designed as a gas fired furnace containing one or more vertical retorts, the contemplated dimensions of which were 3 m. height, 0·3 m. width (outside) at the bottom and 0·2 to 0·25 m. at the top, and 0·4 to 0·5 m. length. If the distillation products were drawn off from both the narrow sides it was considered that the length might be 0·75 to 0·8 m. The bottom of the retort was arranged with a taphole whence reduced lead, matte and slag could be drawn off. Condensers for zinc vapor were connected with the side of the retort. Ore, mixed with coal, was to be charged in at the top, and the non-volatile residuum was to be fused. It was thus contemplated to conduct the distillation and smelting continuously.

[1] German patent, No. 40,768.

Silesian Furnaces.

Direct Fired.—The original Silesian furnaces partook largely of the nature of glass-smelting furnaces, in which respect they bore a strong resemblance to the English furnaces, which is not surprising inasmuch as the process of zinc smelting was introduced in Upper Silesia from England. So far as I know, there are no existing drawings of the original Silesian furnaces, but I conceive that they must have been similar to the English furnaces which are described by Percy, in view of the fact that Ruhberg in his first experiments[1] utilized a wood fired circular glass furnace and distilled the ore in pots. Even after muffles were substituted for pots the circular form of the furnace appears to have been retained.[2] The development from these original types, which is commonly referred to as the old Silesian furnace, was in general use in Upper Silesia up to a comparatively recent time.

Old Form Used in Upper Silesia.—The old Silesian furnace consisted of a rectangular combustion chamber, in which 20 muffles, 10 on a side, were placed through arched windows in the side walls around a grate in the center of the hearth. The arrangement is illustrated in the accompanying engravings. The furnaces were commonly built in pairs with chambers between each pair for the calcination of calamine. The flames from the fireplaces played around the muffles and escaped partly into the calamine calcination chambers, partly into the chambers for annealing retorts and remelting spelter, arranged at the opposite end of the furnace, and partly through openings in the roof directly into the furnace house. The furnace had short chimneys over the calamine calcination chambers and retort tempering and zinc remelting ovens, but the combustion products of the fuel (a lean, short-flame coal) escaped largely into the furnace sheds, which according to all accounts were extremely smoky and highly disagreeable places to work in. The condensers were of the old knee form which produced drop zinc. The consumption of coal was very high, one part of zinc requiring 20 parts of coal, and the loss of metal was also high, amounting to 36% or more, yet notwithstanding those disadvantages these furnaces were in use up to about 30 years ago. That the Silesian smelters were able with them to compete with the Rhenish, Westphalian and Belgian, is attributed by Kerl[3] to the low prices of labor and raw material which prevailed

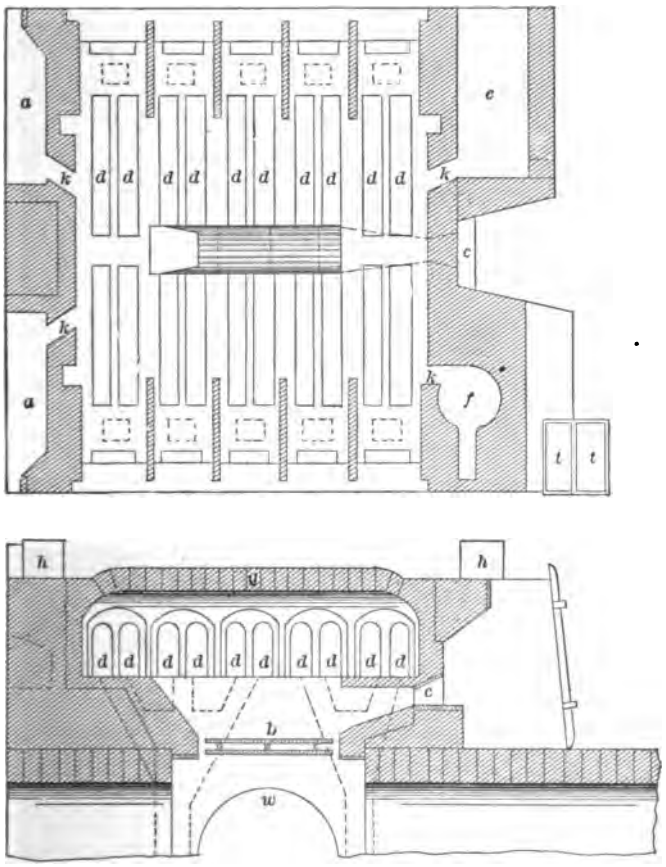
[1] Denkschrift zur Feier des funfzigjährigen Bestehens der Wilhelminezinkhütte, p. 5.

[2] Ibid, p. 7. The muffles used previous to 1820 were about 4 ft. long and 12 to 15 in. width and height, inside measurements. About that time the width was reduced one half and the muffles assumed more or less their present form, which was an improvement of great importance.

[3] Gundriss der Metallhüttenkunde, p. 458.

in Upper Silesia and the comparatively small first cost of the Silesian works.

In the accompanying engravings, reproduced from Kerl, *aa* are the calamine calcination chambers, placed between a pair of furnaces, and com-



FIGS. 255 AND 256. OLD SILESIAN FURNACE.
Fig. 255: Plan. Fig. 256: Longitudinal vertical section.

municating with the distillation chamber of each by the flues *k*; *g* is the arch of the distillation chamber, 0·89 m. high above the hearth; *b* is the fire-place, 1·66 m. long, 0·39 m. wide at the bottom and 0·47 m. wide at the top, the grate being 0·78 m. below the hearth; *dd*, etc., are 20 muffles of which 12 are small and eight are large, the long muffles being

arranged to overlap the fireplace at the ends as shown in the plan; the muffles were 0·56 m. high and 0·22 m. wide at the bottom and 0·25 m. wide at the top, outside measurements, arranged in pairs in niches 0·73 m. high and 0·65 m. wide and 1·33 m. deep, the niches being built with walls 0·1 m. thick; *c* is the stoke hole, 0·55 m. above the furnace floor, opening into the so-called throat (Schlund) in the distillation chamber; *hh* are small low chimneys; *e* is the muffle-annealing chamber, 1·88 m. long, 0·71 m. wide and 0·42 m. high; *f* is a chamber containing a kettle for remelting the drop zinc; *z* is a tunnel 2·2 m. high, with a cross tunnel *w*; *q* is a tile closing the opening in the muffle through which residues are drawn out; *r o m* is the condenser of which the details are shown in another engraving; *n* is the chamber in which the drop zinc collects, this as well as the front of each niche being closed by a door.

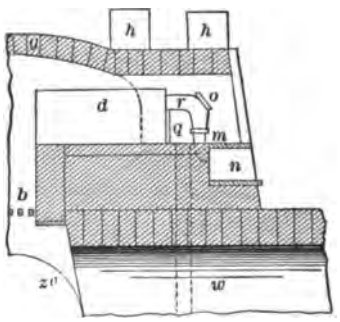
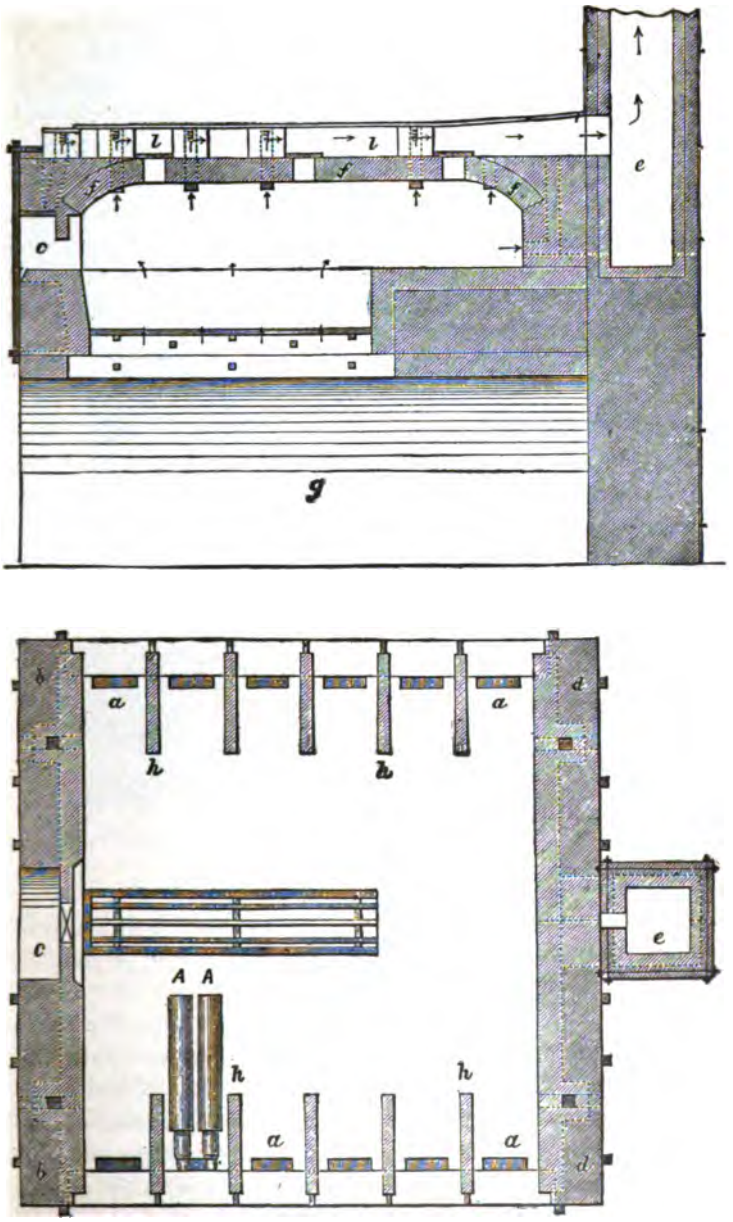


FIG. 257. OLD SILESIAN FURNACE.
Transverse vertical section.

Old Form used in Great Britain.—Furnaces of precisely the same type were employed about the same time in Great Britain, but the fat coal of the latter country permitted good results to be obtained with the stronger draught produced by chimneys. A British-Silesian furnace, used at Llansamlet, Wales, is shown in the accompanying engravings. These furnaces had no accessory chambers for calcining calamine, tempering muffles or remelting zinc. The distillation chamber contained 24 muffles and the fireplace abutted one of the end walls instead of being placed in the middle of the hearth. The products of combustion passed through flues over the muffles into longitudinal flues on top of the furnace and thence to the chimney at the opposite end from the stoking door. Otherwise the design and details of construction were substantially the same as with the furnaces used in Upper Silesia.



FIGS. 258 AND 259. SILESIAN FURNACE USED AT LLANSAMLET, WALES.

Fig. 258: Longitudinal vertical section. Fig. 259: Horizontal section.

Muffles and Condensers.—The muffles and condensers used with these furnaces are shown in the accompanying engraving in which *A* is the muffle, 1·26 m. long, 0·47 m. high and 0·21 m. wide, inside measurements; *n* is a clay plate closing the lower opening in the muffle through which the residues were withdrawn; *oo* are projections, supporting the bridge *b* on which rests the knee-form condenser *p*, attached to which is a cast iron pipe *r*, and to the latter a wrought iron continuation *s*; the condenser *p* is 0·16 m. wide, 0·37 m. high to the drop tube, and 0·47 m. long; the upper part of the cast iron drop tube *r*, is 0·09 m. in diameter, and 0·29 m. long, while the lower part *s*, which is made of sheet iron, is 0·10 m. in diameter and 0·45 m. long;

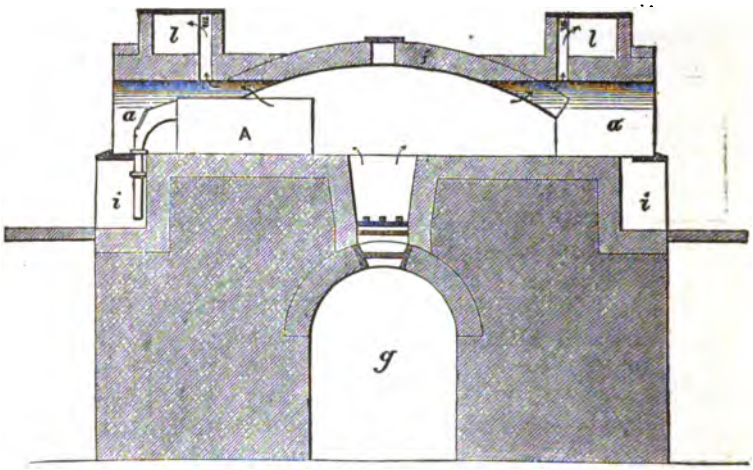


FIG. 260. SILESIAN FURNACE USED AT LLANSAMLET, WALES.
Transverse vertical section

q is a plate closing an opening in the condenser *p* through which tools might be introduced for the purpose of clearing the nozzle from accumulations of oxide and dust. It may be remarked in passing that with the original Silesian furnace, as well as with those of the present time, the retorts were charged through the condensers, which were not removed until the retort itself had to be replaced. The accompanying drawings of the old Silesian retorts and condensers are reproduced from Phillips' *Metallurgy*, while the dimensions are taken from Kerl's *Grundriss der Metallhuettenkunde.*

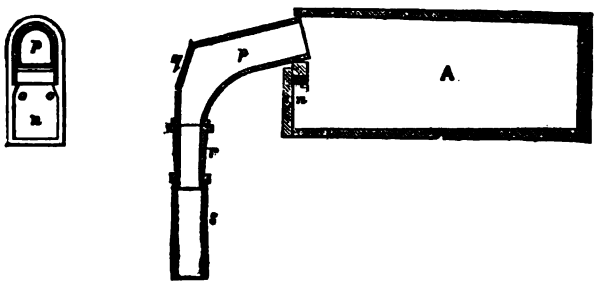
Practical Results.—In Upper Silesia the furnaces with 20 muffles above described used to distill in 24 hours from 750 to 800 kg. of calamine mixed with 2·2 hectoliters[1] of stone coal and 2·2 hectoliters of cinder. Fur-

[1] One hectoliter of coal=76·5 kg. to 80 kg.

naces with 24 and 26 muffles would distil 900 and 1000 kg. of calamine respectively, but the 20 muffle furnaces gave the best results. Muffles lasted from 42 to 56 days. The average yield of zinc was 14·07% of the ore, and to produce 100 kg. of zinc usually required 724 kg. of calamine and 1760 kg. of coal, wherefore one part of zinc produced involved the consumption of 17 parts of coal.[1]

At Swansea a furnace with 24 muffles would distil 711 kg. of roasted blende, 68 kg. of zinc crusts, 254 kg. of coal, and 101 kg. of cinder, or 32·5 kg. of ore per muffle, consuming 1244 kg. of fat coal, 1422 kg. of lean coal, and 254 kg. of cinder; i.e., about 11·5 parts of coal per part of zinc produced. Roasted blende assaying 43·4 units of zinc would yield 36·4 units of spelter, the loss in roasting being 1·5% and in distillation 16%. The campaign of the furnace lasted 13 months.[2]

Modern Forms of Direct Fired Furnaces.—The modern form of the Si-

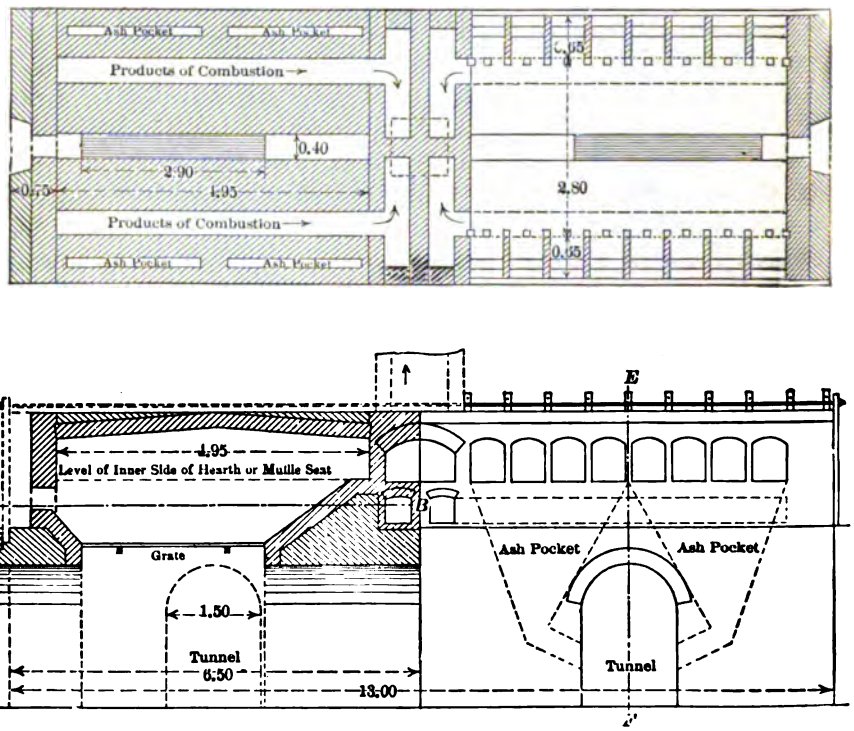


FIGS. 261 AND 262. SILESIAN MUFFLE AND KNEE-FORM CONDENSER.
Fig. 261: Transverse section. Fig. 262: Longitudinal section.

lesian furnace is due to modifications of the original type which were made at the works of the Société Anonyme de la Vieille Montagne at Valentin-Cocq, Belgium, where the Silesian type of furnace was early adopted and was long in use. A fat, long-flaming coal being available at that place, it was possible to conduct the combustion in a way that the Silesians had not previously been able to do with their lean coal. Because of these modifications, the type of furnace, which was also adopted in Silesia, came to be known as the Belgian-Silesian. In this furnace the flames rising from the fireplace to the hearth are drawn downward between the muffles through openings in the hearth on each side and thence into longitudinal flues under the front ends of the muffles, and from the latter flues to the chimney, a central chimney serving a pair of furnaces set end to end. The arrange-

[1] Kerl, Metallurgie, II. 700; Berg- u. Hüttenm. Ztg., 1863, p. 123; ibid, p. 116.
[2] Percy, Metallurgy, first edition, I, 558.

ment is shown in the accompanying engravings. The modified type was adapted to the Silesian conditions by making the fireplace deeper and carrying a bed of clinker about 0·25 m. deep on the grate, thus producing a species of semi-gas firing similar to that which is practiced with the Welsh clinker grate, as has been described in a previous chapter. It became pos-



FIGS. 263 AND 264. FURNACE FORMERLY USED AT VALENTIN-COCQ, BELGIUM.

Fig. 263: Horizontal section on line AB of Fig. 264 and plan of hearth. Fig. 264: Longitudinal section and front elevation. Dimensions are in meters. This furnace is the same as shown in Fig. 265, but engraved on a smaller scale.

sible thus to elongate the flame of the lean Silesian coal so as to obtain the same results as with the more gassy Belgian coal. A further improvement in the firing was the introduction of a blast of air forced by means of a fan or an inspirator, under the grate, the ash pit being of course tightly closed. Other furnaces were run with an upper wind, the air being conducted through flues in the hearth under the muffles and discharged through openings in the fireplace above the bed of fuel.

Besides the improvements in the method of firing, the old knee-form condensers were abandoned in favor of the straight or bellied condensers, such as were used with the Belgian furnace, whereby the necessity for remelting the zinc was obviated, and a less quantity of between product was made, though the spelter was not so pure as the drop zinc. The hearth was given a slight inclination toward the outer sides, so that in the muffles resting

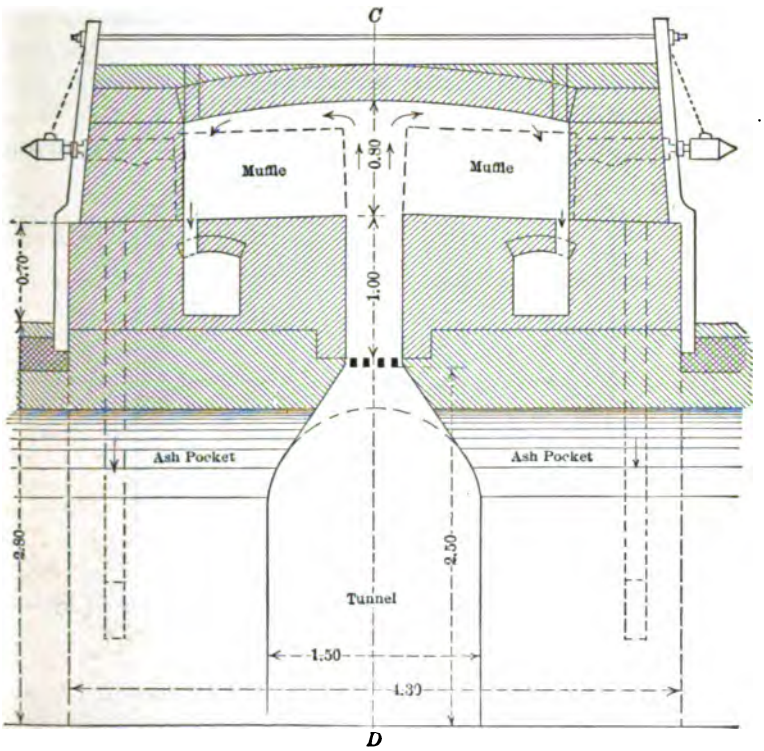


FIG. 265. FURNACE FORMERLY USED AT VALENTIN-COCQ, BELGIUM.
Transverse vertical section on line EF of Fig. 264. Dimensions in meters.

upon it the slags would run toward their outer and cooler end, whereby its corrosive effect was lessened. Furthermore the tunnels under the furnace were arranged so that the residues from the retorts could be dropped into them through holes in the niches, thus permitting of economy in labor and increasing the comfort of the furnacemen.

By virtue of these improvements it was possible to increase the number of retorts in the distillation chamber to 32 (16 per side) or even 40 with

long flaming coal, and at the same time maintain a more even temperature with a smaller consumption of coal, which increased the durability of the retorts and the percentage of zinc extracted, besides the saving in coal, the consumption of which was eventually reduced to six to eight parts per part of zinc. Moreover the products of combustion being discharged through chimneys instead of to a large extent into the furnace houses, the latter became more healthful places in which to work and the efficiency of the men was increased. The calamine calcination chambers arranged between a pair of furnaces of the original type were preserved, but these are not an essential part of the furnace and at the present time some furnaces are to be found with them and some without them.

Owing to the difficulty in regulating the temperature of the condensers in

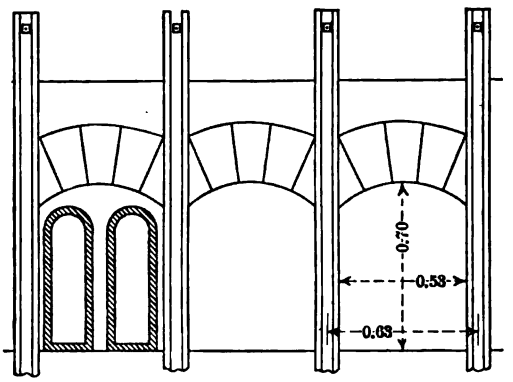


FIG. 266. FURNACE FORMERLY USED AT VALENTIN-COCQ, BELGIUM.
Elevation of niches. Dimensions in meters.

the Silesian furnaces where two condensers, each corresponding to a retort in the furnace, are contained in a niche (Kapelle), separated from each other by brick walls, wherefore those muffles first charged would be hotter than those muffles last charged, at some works the solid partition walls of the Kapellen were abandoned, open castings or iron plates pierced with holes being substituted for them. The condensers of each side of the furnace were contained thereby in practically one large chamber, instead of in a number of small ones, and an even temperature could be maintained therein. This was found to be better than to attempt to regulate the temperature of each pair of condensers, a task that is indeed beyond the capacity of the ordinary workman, although the temperature in the large chamber might not correspond to the requirements of the condensers in different parts of

the furnace.[1] Cochlovius patented an open, cast iron supporting frame to replace the clay plates ordinarily used between the niches (vide Fig. 275).[2]

The general features of the construction of a direct fired Silesian furnace will be understood without further description from the accompanying engravings, which show a furnace essentially like those formerly used at Valentin-Cocq, Belgium. The furnaces used in Upper Silesia are the same, save that they are sufficiently wider to receive muffles about 1·7 m. long. Two furnaces are commonly combined in a block, with muffle tempering chambers between them, and a central chimney.

NUMBER AND KINDS OF DISTILLATION FURNACES IN USE IN UPPER SILESIA.

Year	1891						1901					
Kind of furnace	Direct		Gas		Siemens		Direct		Gas		Siemens	
Name of works	N	M	N	M	N	M	N	M	N	M	N	M
Bernhardi											8	640
Beuthener	8	228							*d*16	*d*676		
Carls			10	320							10	320
Clara	12	288					12	288				
Fanny Franz	26	720					26	720				
Flora									*e*6	*e*216		
Franz			*a*8	*a*272							8	256
Kgl. Friedrichs					2	48					2	48
Godulla			*b*12	*b*748	*b*8	*b*500					*c*20	*c*2,720
Guidotto			16	512					30	1,758		
Hohenlohe			72	2,304					72	2,304		
Hugo			18	672							*c*20	*c*2,400
Kunigunde	*b*3	*b*192			2	120	10	368				
Lazy			20	704							*f*20	*f*960
Liebehoffnungs			44	1,472					40	2,214		
Lydognia	13	348										
Norma					4	272					4	272
Pauls	16	512			12	776	16	512			12	792
Rosamunde	8	284					*d*	*d*	*d*	*d*	*d*	*d*
Silesia II			60	1,976					60	1,978		
Silesia III			56	1,880					60	1,984		
Theresia	8	264					10	324				
Thurso	10	240							10	280		
Wilhelmine					30	2,028					30	2,028
Totals	104	3,096	316	10,860	58	3,744	74	2,212	294	11,410	134	10,436

a Semi-gas fired. *b* Double furnaces. *c* Double furnaces with two or three rows of muffles. *d* The statistics of the Beuthener and Rosamundehütte are combined. *e* Recuperative furnaces. *f* Furnaces with two rows of muffles. *N* Number of furnaces. *M* Aggregate number of muffles.

GAS FIRED.—At the present time most of the distillation furnaces in use in Upper Silesia are gas fired. The simple gas fired furnaces differ from the direct fired only by dispensing with the fireplaces and introducing the combustible in gaseous form from a producer built usually at one side of

[1] Denkschrift zur Feier des fünfzigjährigen Bestehens der Wilhelminezinkhütte, p. 82.

[2] Dingler's Polytech. Journ., CCXXXVII, 301; German patent, No. 9128.

the furnace. Gas firing has made it possible, however, to employ efficient systems of heat recuperation, and through the feasibility of introducing the gas at numerous points and of elongating the flame to almost any extent desired the distillation chamber has been increased greatly in size, so that the latest furnaces at the Bernhardihütte of G. von Giesche's Erben, a company which has always been in the lead in the utilization of gas fired regenerative furnaces, have 80 muffles, 40 per side, the largest previously in use in Upper Silesia having only 72. The Bernhardihütte was built in 1897 and 1898. The number and types of distillation furnaces in use in Upper Silesia at the end of 1891 and 1901 are given in the foregoing table which shows the increasing tendency toward the employment of gas firing.

The foregoing table, which is compiled from the *Statistik der Oberschles-*

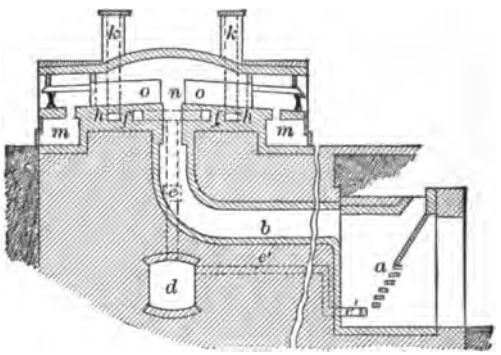


FIG. 267. GAS-FIRED SILESIAN FURNACE.
Vertical section on line AB of Fig. 268.

ischen Berg- und Hüttenwerke, although probably containing some errors, is nevertheless valuable as indicating the general tendency of the decade toward Siemens furnaces and recently the adoption by several works of furnaces with two or three rows of small muffles. Of the old type of furnaces, including direct fired and gas fired of all kinds, the number of muffles per furnace in some of the works was as follows: Bernhardi, 80; Carls, 32; Clara, 24; Flora, 36; Franz, 32; Hohenlohehütte, 32; Norma, 68; Pauls, 32 (direct fired) and 60 to 72 (gas fired); Thurzo, 28. Of the furnaces with two or three rows of muffles, those at the Godullahütte have 136; at the Lazyhütte, 48; at the Hugohütte, 120. A new plant now in course of construction at Hohenlohehütte, to be completed about the end of 1903, is to have six double furnaces, each with 192 muffles arranged in two rows.

Ordinary Type.—A gas fired Silesian furnace with 48 retorts is shown in Figs. 267 and 268. This furnace has two heating shafts and two step-

grate gas producers built opposite to one of the long sides. *AA* are the producers, which are blown under the grate with air from the canal *d*. The gas enters the furnace through the canals *bb* and rises through the flues *cc*. The latter are surrounded by air flues *ff*, which discharge air through the openings into the stream of gas near the top of the flues *cc*. The air for this secondary combustion comes through the main flue *d*, rises through the vertical flue *e* at the end of the furnace, whence it is led back through the horizontal flues *ff* under the hearth of the combustion chamber. The products of combustion pass downward through the flues *hh* between each pair of muffles into the flues *h'h'*, whence they rise through the small

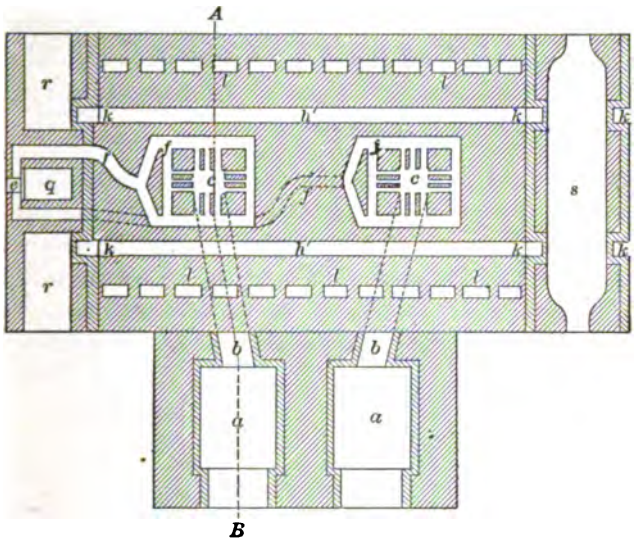


FIG. 268. GAS-FIRED SILESIAN FURNACE.
Horizontal section.

chimneys *kk* and the main chimney *q*. *S* is a chamber for the calcination of calamine between a pair of furnaces and *rr* are rooms for annealing retorts, each of these being so arranged with flues that a part of the products of combustion can be diverted through them. *Oo* are muffles, *ll* are the chutes through which the residues from the retorts are dropped, and *mm* are the pockets which receive them. This was one of the earliest forms of gas fired furnaces employed in Upper Silesia. It was built with one gas flue and 20 to 28 retorts and two gas flues with 40 to 56 retorts, the latter type being preferred because of economy in fuel and labor and comparatively less first cost.[1]

[1] Kerl, Grundriss der Metallhüttenkunde, p. 465.

Furnaces at Hohenlohehütte.—A departure from the characteristic Silesian system of heating the retorts appears in a 64 muffle furnace used at the Hohenlohehütte, which is shown in the accompanying engravings. This furnace, which is 16·1 m. long and 5·62 m. wide, has two step-grate gas producers and four gas flues *GGGG*. The air for the secondary combus-

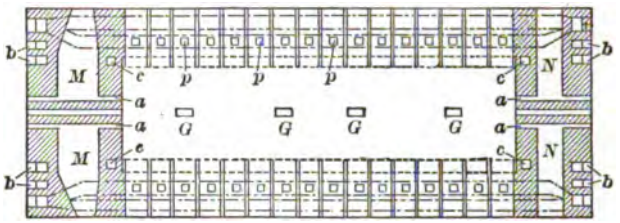


FIG. 269. FURNACE USED AT HOHENLOHEHÜTTE.
Horizontal section.

tion is blown by a fan through the main canal, from which branches convey it around the gas flues *G*, finally mixing with the gas by passing through the ports shown in the vertical section. , The products of combustion instead of drawing downward between the retorts into flues under the hearth, are drawn to the ends of the combustion chamber and thence through the flues *aa* into the chambers *MM* for annealing muffles and *NN*

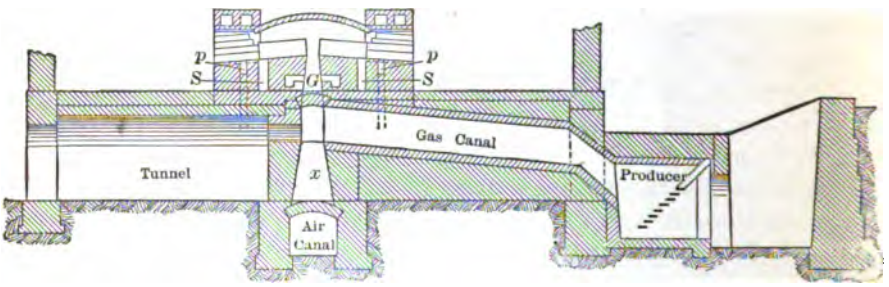


FIG. 270. FURNACE USED AT HOHENLOHEHÜTTE.
Transverse vertical section.

for calcining calamine, whence they are led through the flues *bb* into the chimney. Such part of the waste gases as is not required for those purposes is led directly to the chimney through the flues *cc*; *x* is a chamber for the collection of dust and soot deposited from the gas and *ppp* are chutes for the discharge of retort residues. Access is obtained through the tunnel as shown. *SS* are chambers for the accumulation of slag fused from the out-

side of the muffles. The cross-section shows the retorts equipped with Dagner condensers and the collecting gas flues over the longitudinal walls of the furnace.

Haupt Furnace.—This is a gas fired Silesian furnace, designed by C. Haupt, with recuperative chambers for preheating the air for secondary combustion.[1] On each side of the long axis of the furnace, beneath the hearth, there is a series of recuperative chambers, each of which is filled with brick in such a way as to divide it into a series of horizontal flues alternating with vertical flues. The air for combustion of the gas passes through the horizontal flues and meets the gas in the shafts midway in the furnace. The burning gas rises to the arch of the furnace and passes down between the muffles and escapes through ports in the hearth in the usual manner. The products of combustion before reaching the main flue leading to the chimney have to go through the vertical flues of the recuperative chambers, wherein they give up part of their heat to the air passing through the horizontal flues.

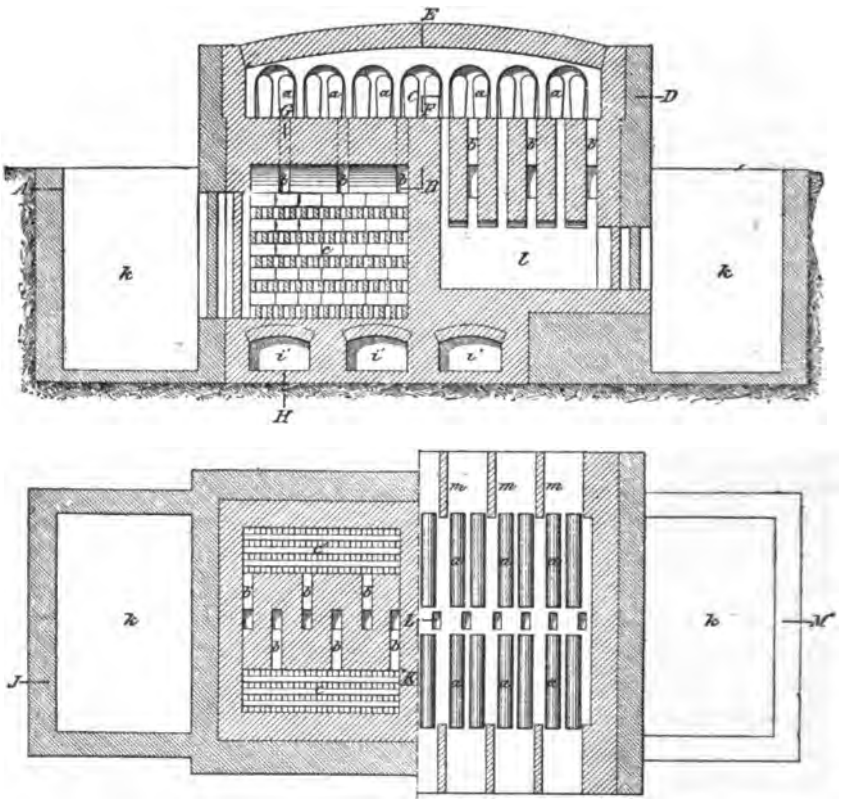
According to Kosmann,[2] the Haupt furnace was in use for a time at the Godullahütte, where it showed a saving of 25% in fuel (it is not stated with what the comparison was made, but presumably direct firing is meant). The furnace was abandoned, however, because certain inaccessible parts were subject to rapid destruction. The Haupt furnace is interesting as an application of the counter-current system of heat recuperation to the characteristic Silesian type, although it does not appear to have been well developed.

Siemens-Silesian Furnaces.—The application of the Siemens system of heat recuperation to the Silesian distillation furnace involves still a different method of heating the retorts, since the products of combustion must not only be made to travel horizontally, but their direction is periodically reversed. The principle is shown in the accompanying engravings of a 28 retort furnace at Freiberg, Saxony. This furnace it should be noticed has a dome roof instead of a simple arch. The regenerative chambers containing a checker-work of fire brick are built under the hearth of the combustion chamber and parallel with the long axis of the furnace. There are two regenerative chambers, one for gas and one for air, divided by a longitudinal middle wall, with flues into which each chamber communicates. Each chamber is further subdivided into halves by a transverse wall. Under each chamber there are two flues, one of which communicates with the near

[1] German patent, 7425; Oest. Zts. XXIX, 336; Kosmann, Verhandl. des Vereins zur Befoerderung des Gewerbefleisses, 1881, p. 182.

[2] Oberschlesien, sein Land und seine Industrie, p. 204.

section and the other with the far section. These flues are shown as *i'i'i'* in Fig. 271. The flues connecting the regenerative chambers with the combustion chamber are shown as *b* and *b'*. It will be observed that there is alternately an air flue and a gas flue, three of each opening into each half of the combustion chamber.



FIGS. 271 AND 272. SIEMENS-SILESIAN FURNACE; USED AT FREIBERG.
Fig. 271: Plan. Fig. 272: Longitudinal vertical section.
Scale, 1:100.

In operation, gas and air are turned into the flues leading to the near halves of the respective regenerative chambers. The gas and air pass through the checker-works and the flues *b* and *b'* into the combustion chamber, where they burn, playing around the retorts, and draw into the other half of the chamber, pass through the flues *b* and *b'* in that half, through the checker-work and thence through the flues *i'i'* corresponding to those

chambers to the chimney. At the end of 30 to 60 minutes valves in the gas and air canals are reversed so that the incoming air and gas are led

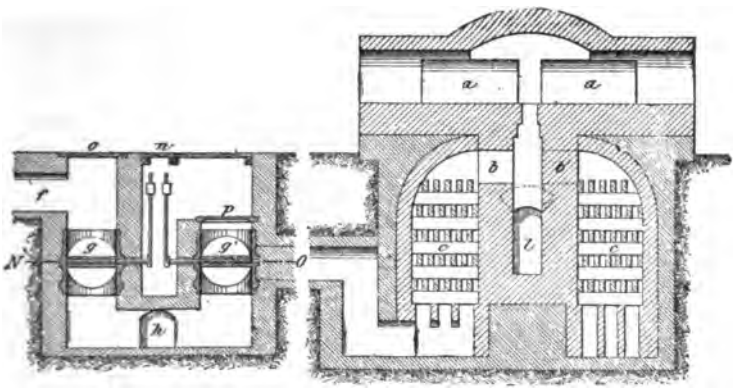


FIG. 273. SIEMENS-SILESIAN FURNACE, USED AT FREIBERG.
Transverse vertical section.
Scale, 1:100.

through the flues and checker-works through which the products of combustion were previously escaping, while in the combustion chamber the flames travel now in the opposite direction and the products of combustion

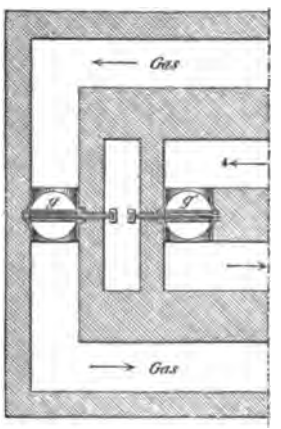


FIG. 274. SIEMENS-SILESIAN FURNACE, USED AT FREIBERG.
Arrangement of gas and air reversing valves.
Scale, 1:100.

escape through the ports which previously turned in the fresh air and gas. The currents are thus reversed periodically and the gas and air for com-

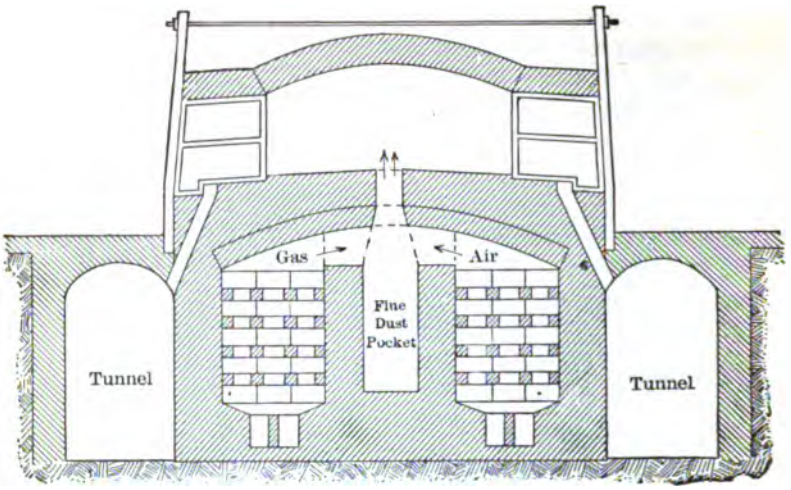


FIG. 275. SILESIAN-SIEMENS FURNACE.

Transverse vertical section, showing Cochlovius iron frames, separating the niches.

bustion are heated by passing through the hot checker-work. The accompanying engravings will be perfectly clear with the above explanation. It is only necessary to remark that *l* is a chamber for the deposition of dust, soot, etc., from the fuel gas. The system of reversing valves will be understood as shown.

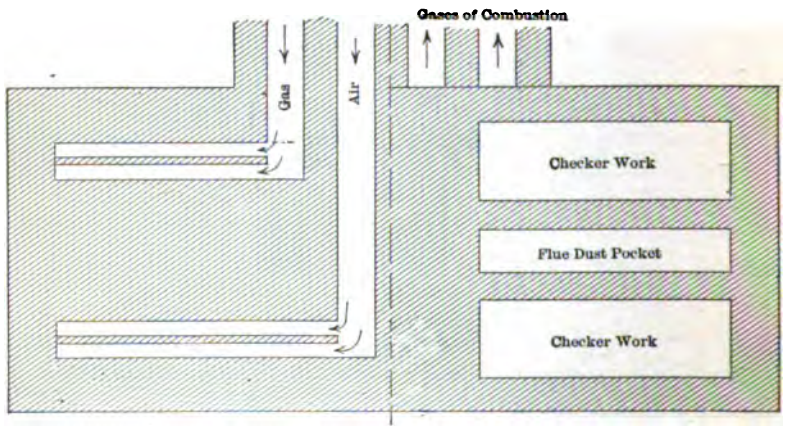


FIG. 276. SILESIAN-SIEMENS FURNACE.

Horizontal sections through gas and air canals, and through regenerative chambers. This engraving is on a slightly smaller scale than that of Fig. 275, to which, however, it refers.

The general arrangement of the Siemens furnaces employed in Upper Silesia are illustrated in Figs. 275 and 276, of which the former shows the Cochlovius partitions dividing the niches. The gas flues enter at right angles to the longer axis of the furnace, the producer being outside of the furnace house.

The relative economy of direct fired furnaces and Siemens regenerative gas fired furnaces is illustrated by the experience at the Paulshütte near Rosdzin, as reported by Schnabel in *Handbuch der Metallhüttenkunde,* II 176. The Siemens furnaces in use there have 60 to 72 muffles, which are 1·7 to 1·8 m. long. The muffles are made of a mixture of 65% clay and 35% old material. The charge consists of 30% roasted blende and 70% calamine, mixed with cinder to the amount of 40% of the weight of the ore. The charge for a muffle weighs 100 to 110 kg. The life of the muffles is 40 to 50 days. Each furnace has two gas producers. A 60-muffle furnace reduces 6000 kg. of ore per day and yields about 1000 kg. of zinc with a consumption of 6100 kg. of heating coal, i.e., 1·016 ton of coal per ton of ore=6·1 ton of coal per ton of zinc. The consumption of muffles amounts to 1·35 per ton of zinc. The yield of zinc is 13% of the weight of the ore. The Siemens furnaces do not require any better coal than direct fired furnaces with undergrate blast of air, but consume less and work more evenly. The comparative results obtained from the direct fired furnaces and the Siemens furnaces in 1895 are given in the following table:

Data	32-muffle direct-fired furnace	60-muffle Siemens furnace
Ore smelted per day	3,000 kg.	6,000 kg.
Consumption of heating coal, per 1,000 kg. of ore	1,330 "	1,016 "
Breakage of muffles, per 5,000 kg. of ore	1·6	1
Yield of zinc, per ton of ore	13%	13%

Rhenish Furnaces.

The type of distillation furnace which is employed generally in Rhenish Prussia and Westphalia, and to a considerable extent in Belgium, and for convenience is referred to in this treatise as the "Rhenish," is a direct development of the Silesian.[1] There is indeed no other distinctive difference that can be specified besides the one that it is now commonly built with three rows of retorts, which are smaller than those used in Upper

[1] The designation of this type of furnace as the "Rhenish" has been adopted in order to avoid the confusion that is attached to the terms "Belgian-Silesian" and "Silesian-Belgian."

Silesia. The great width of the Silesian furnace, with the deep niches and broad arched roof of the latter are preserved, as is also the characteristic up and down draught, but in height of the combustion chamber and arrangement of the retorts there is resemblance to the Belgian type. The first Rhenish furnaces had only two rows of retorts and they were of muffle shape like the Silesian, but smaller. The influence of the Belgian smelters led to the introduction of a third row of retorts, and their shape was more or less modified, approaching still more to the Liège type.

As long as 16 years ago furnaces were built in Upper Silesia with two rows of small muffles, repeating the first step in the evolution of the Rhenish type, but at first they did not meet with favor.[1] In these furnaces, as in the original Rhenish furnaces, the lower row of retorts rested directly on the hearth, only the upper row being supported on shelves. In the modern Rhenish furnace the lower row is supported above the hearth so that the flames can envelop its retorts on all sides, as in the Belgian furnace. The opinion has been repeatedly expressed by Silesian metallurgists that as the percentage of roasted blende to be treated should increase there would be a tendency toward the adoption of the Rhenish furnace in that district, and this forecast now seems to be materializing, the Hugohütte having installed furnaces of the Rhenish type in 1897, since which time furnaces with two rows of muffles have been adopted by several other works.

The Rhenish furnaces which are used in Belgium are either direct fired or gas fired; those which are used in Rhenish Prussia and Westphalia are chiefly gas fired, commonly with Boetius generators built integrally with the furnace; some furnaces have two producers at each end of the massive, while other furnaces have only one at each end. One of the earliest uses to which the Boetius generator was applied was the heating of zinc distillation furnaces and it has ever since been a favorite for that purpose. The choice between direct firing and gas firing depends a good deal upon the character of the coal available. In Belgium, where the domestic coal is only semi-fat, direct firing is still largely continued, such works as have adopted gas firing being obliged to import German coal. It having been always the policy of the Société Anonyme de la Vieille Montagne to use the coal most easily obtained,[2] direct firing is retained at its works at Valentin-Cocq (the largest single producer of zinc in the world)

[1] The Guidottohütte erected at Chropaczow in 1887 was equipped with eight double furnaces, each with 128 muffles (four in a niche) arranged in two rows, the furnaces being similar to what were in use at Dortmund. According to Kosmann (Oberschlesien, sein Land und seine Industrie, p. 211) this was the first experiment with this type of furnace in Silesia. The results were said to have been unsatisfactory. Similar furnaces were in use experimentally at the Hohenlohehütte in 1893.

[2] De Sinçay, The Mineral Industry, VIII, 655.

and at Flône, where Rhenish furnaces are used exclusively, although at Angleur, where all the furnaces are of the Belgian type, gas firing is employed. It is important to remark that at other Belgian works, especially those at Overpelt, Prayon and Engis the tendency is toward the adoption of Siemens furnaces with three rows of large retorts of about the same dimensions as those used in Rhenish Prussia.

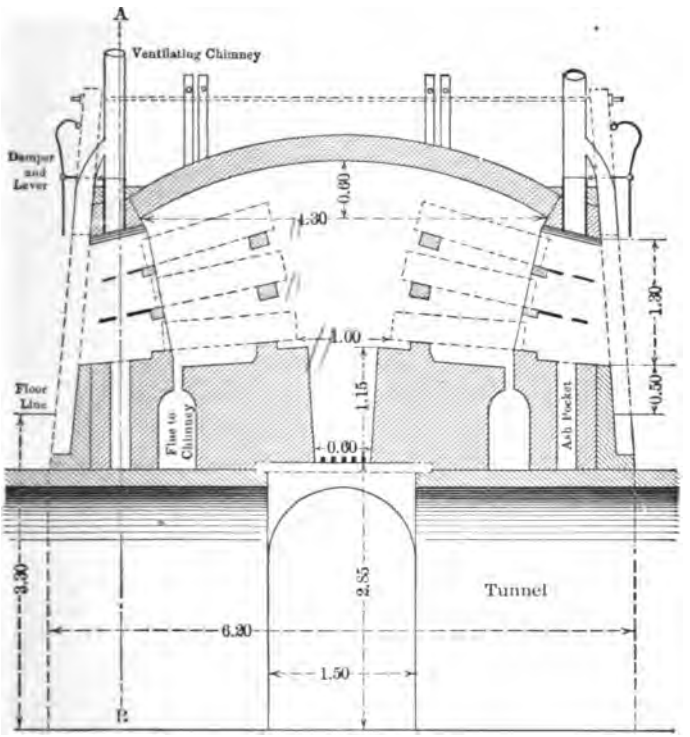


FIG. 277. FURNACE USED AT VALENTIN-COCQ, BELGIUM.
Transverse vertical section. Dimensions are in meters.

DIRECT FIRED.—The furnaces of most recent construction at Valentin-Cocq are illustrated in the accompanying engravings, which are redrawn from a plate in a paper by Ad. Firket.[1] Like all the furnaces at Valentin-Cocq they have 108 retorts arranged in three rows of 18 each per side, thus giving 54 retorts per side. The retorts are elliptical, 0·168 m. wide and 0·254 m. high, inside measurements, and 1·40 m. long. They are heated

[1] Annales des Mines de Belgique, VI. ii.

by a longitudinal grate, midway in the combustion chamber, whence the flames rise to the roof arch and thence draw down between the retorts, escaping through ports near the faces of the furnace, just as in the conventional, direct fired Silesian furnace. Each face is divided vertically into nine niches, of which each contains six retorts. The niches are divided horizontally by cast iron plates; each plate has an opening for the descent of the ashes when the retorts are cleaned out, the openings being closed at other times. The fronts of the niches can be closed by means of sheet iron plates

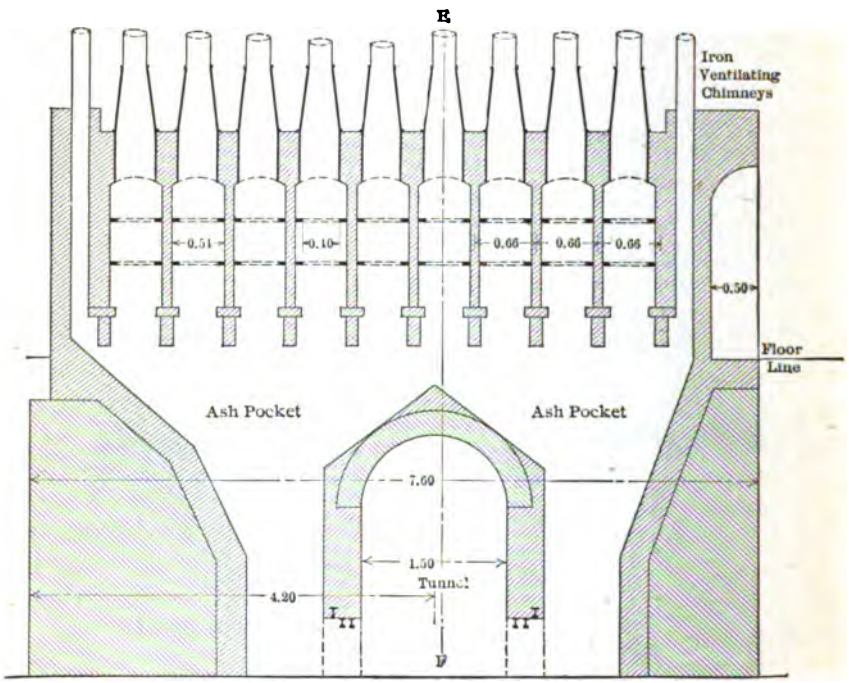


FIG. 278. FURNACE USED AT VALENTIN-COCQ, BELGIUM.
Longitudinal section through niches on line AB of Fig. 277, showing ventilating chimneys. Dimensions are in meters.

and the men are thereby shielded from the heat of adjacent retorts while at the work of discharging any particular one. Two furnaces are joined end to end to form a massive, which is about 15 m. long. At Valentin-Cocq they are arranged in several large buildings on the line of their longitudinal axis, with a space of 3 m. between adjacent massives and 4 m. to 6 m. space between the wall of the building and the faces of the furnaces.

The accompanying engraving shows a furnace equipped with ventilators

to preserve the health of the furnacemen by removing from the building the gases discharged from the furnace and the dust which is raised especially in removing the residuum from the retorts. Above each niche there is placed a sheet iron chimney, which rises to the height of the top of the roof of the building, or about 11 m. above the floor. In front of each niche, at the top of the opening, there is a small hood connected with the chimney which removes the gases issuing from the condensers and prolongs during the distillation. The main chimney is closed below the branch pipe by means of a gate, which is opened only when the residues are to be removed from the retorts. At each corner of the furnace there is an additional chimney to remove the fumes arising from the residues while cooling in the ash pockets. The arrangement has been so satisfactory that all the furnaces at Valentin-Cocq are to be equipped in the same manner, although the cost is about 2000 fr. ($400) per furnace.

The furnaces at Flône are of the same general type as those at Valentin-Cocq, but the retorts are smaller, being only 0·16×0·23×1·34 m. At the end of 1898 Valentin-Cocq had 69 furnaces of 108 retorts each, and Flône 24 of the same number of retorts. The production of spelter by the two works in 1898 was 24,397 and 9110 metric tons respectively.

Gas Fired, Non-Recuperative Furnaces.—A modern Rhenish furnace with 216 elliptical retorts, arranged 108 per side in three rows of 36, is shown in the accompanying engravings. There is a Boetius generator at each end. The air of secondary combustion is warmed in flues surrounding the generator and joins the gas through ports in the gas flue. The products of combustion envelops the retorts and escape through flues at the sides of the hearth into the longitudinal canals which debouch into the main flue leading to the chimney. The front walls are built of refractory tiling, dividing them into niches, in the usual manner. The rear ends of the retorts rest on shelves, corresponding to those of the front walls, of which the construction is clearly shown in the engravings. The residues raked out of the retorts accumulate in the pockets on the ground level, whence they have to be shoveled—an arrangement that could easily be improved. Hoods are provided, above the fronts of the furnace, to remove fumes and noxious gases, as is common in many of the modern Rhenish and Belgian furnaces.

The two gas producers have grates 38×50 in., giving a total grate area of 26·4 sq. ft. The combustion chamber of the furnace is 10 ft. wide and 41 ft. long, its volume being 2829 cu. ft. The arch has a rise of 12 in. The main flue to the chimney is 14 sq. ft. in section.

According to Dürre, to whom the accompanying engravings are due, this

furnace is one of the most approved designs in Rhenish Prussia and as a type has given the best results in zinc smelting.[1]

The distillation furnaces which are used at Dortmund, Westphalia, have 240 retorts, arranged in three rows of 40 each per side. The retorts are

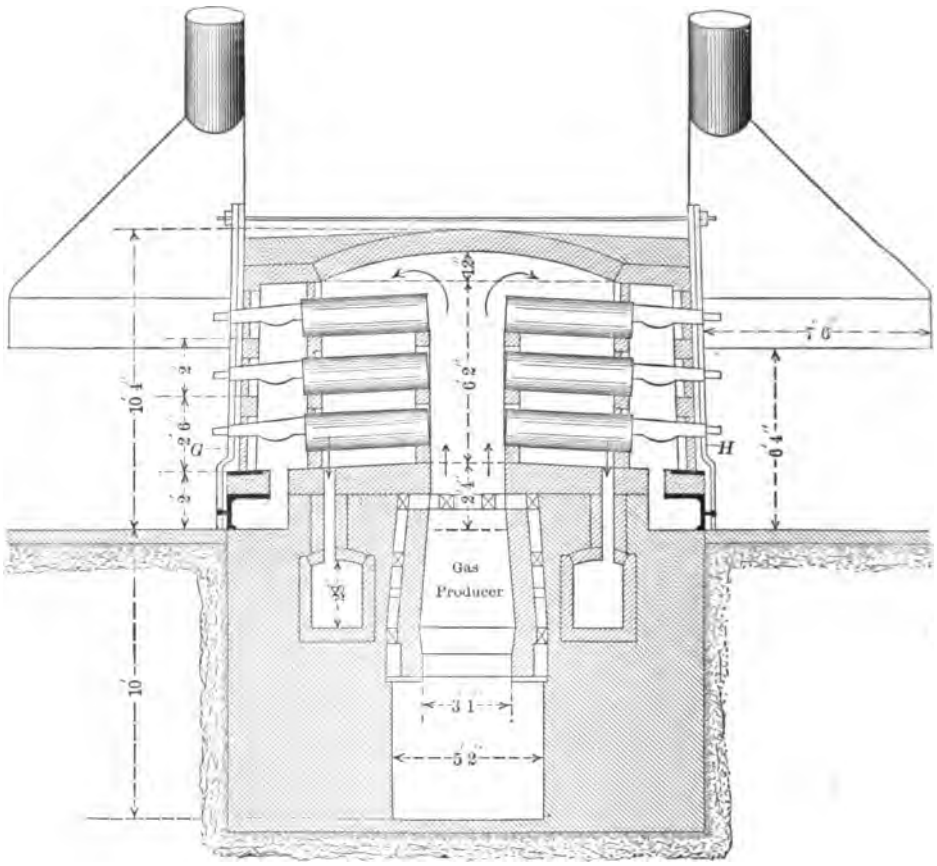


FIG. 279. RHENISH FURNACE.
Transverse section on line CD of Fig. 281.
Scale, 1 : 80.

of 55 cu. decimeters capacity each, and the charge of roasted ore for the 240 is 8000 kg. per 24 hours. At each end of the massive there are two gas producers, which set below the floor level of the furnace house. The air of secondary combustion circulates through flues around the fire box of

[1] Ziele und Grenzen der Elektrometallurgie, p. 207.

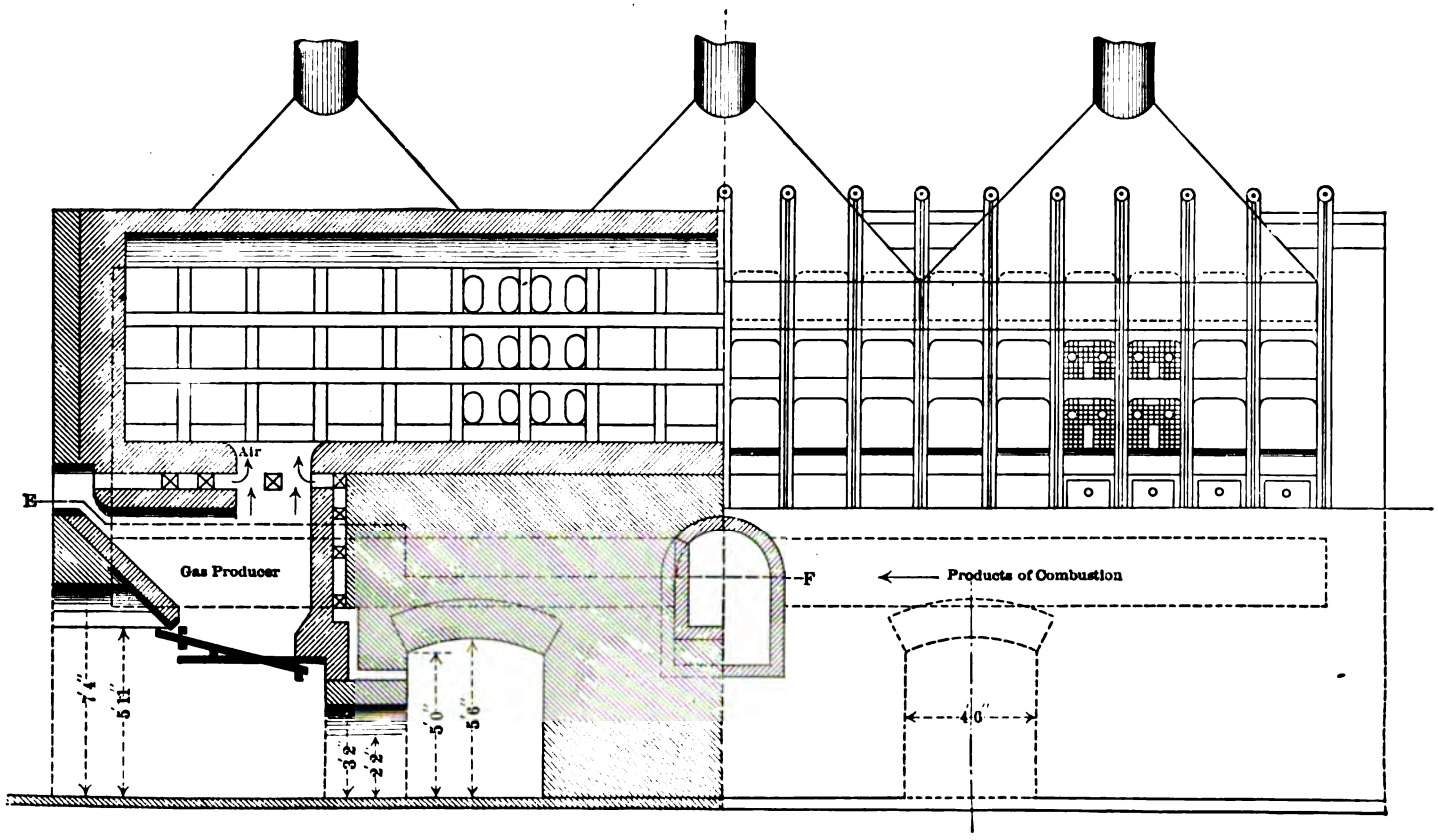


FIG. 280. RHENISH FURNACE.
Longitudinal section and front elevation.
Scale, 1 : 80.

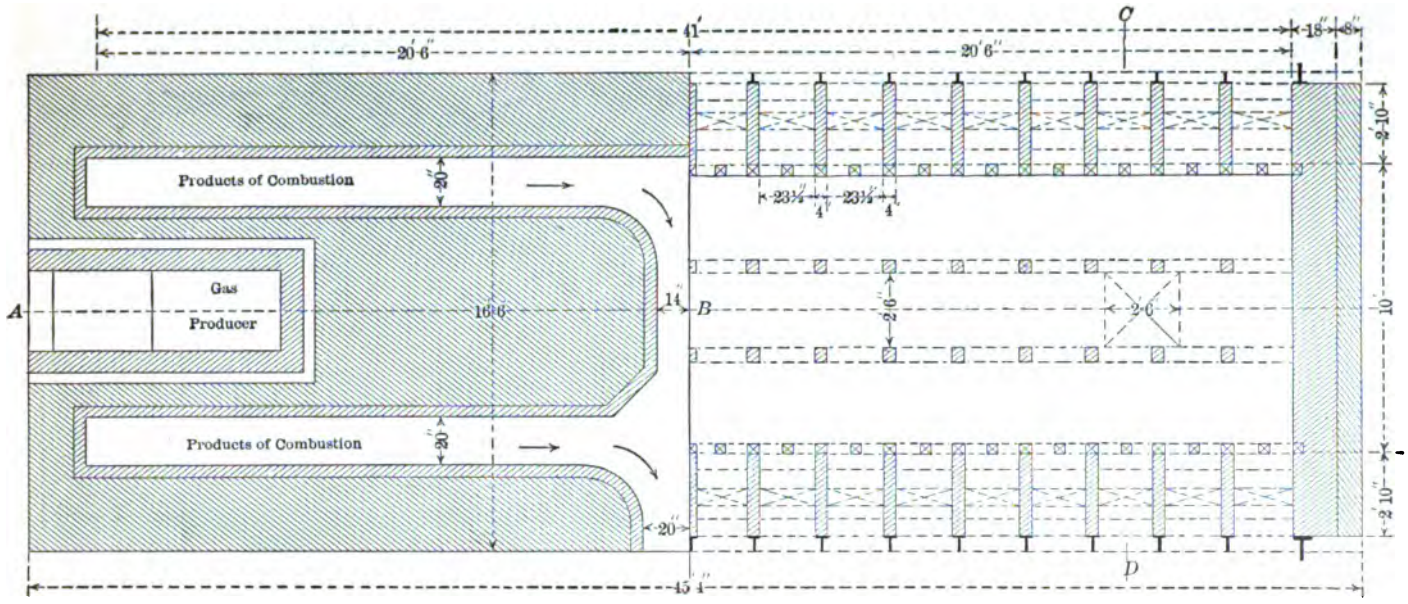


FIG. 281. RHENISH FURNACE.

Horizontal sections EF on Fig. 280 and GH on Fig. 279.
Scale, 1 : 80.

the producer, thence through flues in the end wall of the massive, and finally passes into a rectangular flue above the arch of the latter, from which it enters the combustion chamber through ports above each double tier of retorts. Mixing with the gas, the flames in descending envelop the retorts and pass out through ports in the hearth into flues which conduct the gases of combustion to the chimney. The retorts are rectangular in cross-section, with rounded corners. The niches are separated by cast iron standards and can be closed in front by means of iron plates to regulate the temperature. The fumes arising during the maneuver and the distillation are removed by hoods, leading to sheet iron chimneys which rise above the roof of the furnace house. There are 20 hoods to a massive, or 10 per side. Five massives are grouped in each house.[1]

GAS FIRED, RECUPERATIVE FURNACES.—The Rhenish metallurgists who have built regenerative furnaces have employed the continuous counter-current system rather than the Siemens alternating reversing system. The regenerators of the former type are designed on the principle of those employed many years ago by Gaillard & Haillot and elaborated by Lenchauchez by whose name it is frequently referred to.[2] The systems of C. Nehse and others are merely modifications of that type. In all such recuperators the products of combustion are led around thin-walled fire clay flues through which the air for secondary combustion is forced or drawn; or the products of combustion are led through such flues and the air around them. Fire clay being a poor conductor of heat the walls of the flues should be as thin as possible. At the best a heat recuperator of this system cannot be so efficient as one of the Siemens system, but it is less troublesome to operate, involving no elaborate system of reversing valves. At certain works in Belgium the most recent constructions have been Siemens furnaces,[3] which although differing in various particulars from the standard Rhenish type embody so many of its characteristics as to be best classified therewith.

Counter-Current System.—A Rhenish recuperative furnace of old design, used at Bergisch-Gladbach, is shown in Fig. 282. Under the hearth, which is thick enough to contain the air and gas flues, there are two large chambers in which are placed the recuperative flues, the latter being rectangular fire clay pipes molded with a middle diaphragm. These flues are supported by a series of light brick walls, parallel with the plane of the section, which divide the chambers into numerous rooms, connected, however, by openings, *G*. The ends of the pipe sections break joint with the walls and the space

[1] Rev. Universelle des Mines, 1894, XXV, 38.

[2] A. Lenchauchez, Etude sur les Combustibles en général et sur leur Emploi au Chauffage par les Gaz, Paris, 1878.

[3] It is noteworthy that the furnaces at these Belgian works have been built by German engineers.

between the parallel pipes being bricked up the flues are gas tight so long as they remain uncracked. The gas enters through the canal *A*, from which it passes to the combustion chamber through flues *B*, spaced at proper intervals. The air for secondary combustion enters the lower half of the heating flues through the ports *eee*, which are provided with dampers *fff* for regulation of the volume. The air flows through the lower flues to the opposite

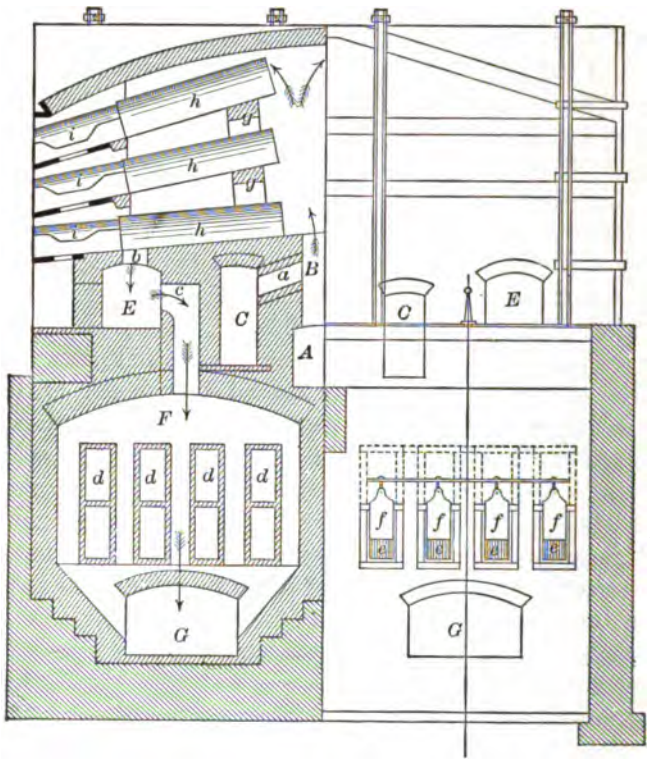


FIG. 282. RHENISH RECUPERATIVE FURNACE.
Transverse vertical section and end elevation.

end of the furnace, where it enters the upper flues and comes back in the opposite direction. The ends of the upper flues terminate in a chamber which communicates with the longitudinal flue *C* in the hearth of the furnace. From the latter the branch flues *a* conduct it to the gas. The products of combustion rise to the arch of the distillation chamber, as shown by the arrows, and coming down between the retorts escape through the ports *b*

into the longitudinal canal *E*. From the latter there are flues *c* corresponding to each transverse division in the recuperative chamber. The hot gases pass thence around the air flues and are drawn through the openings *G* into a flue extending beyond the end wall of the furnace and connecting with the chimney.

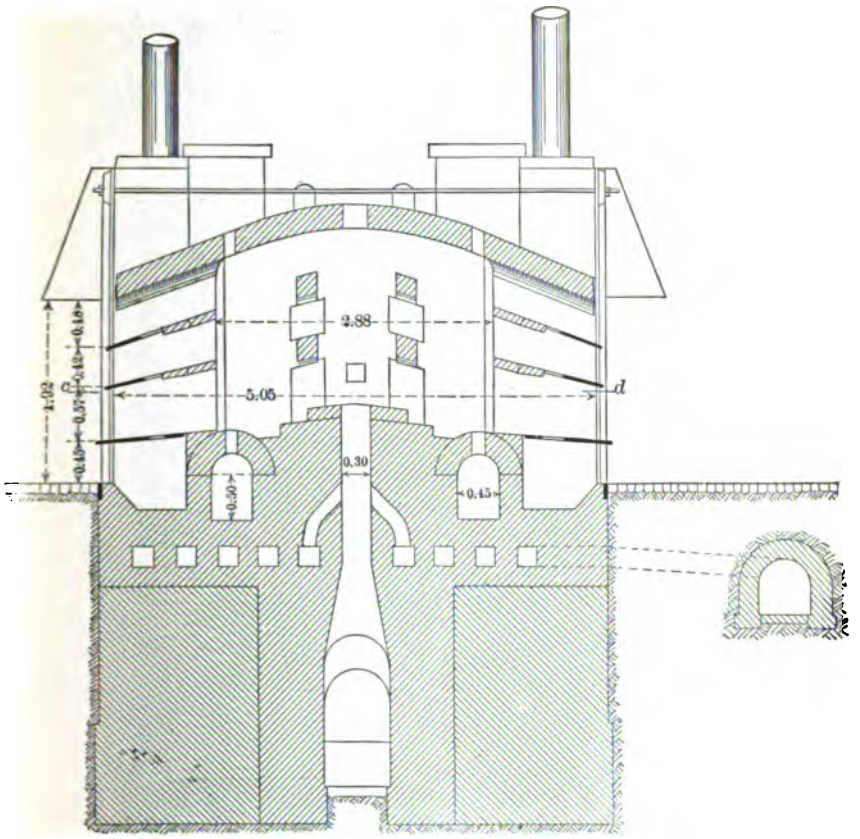


FIG. 283. GAS-FIRED RHENISH FURNACE.

Vertical section on line ab of Fig. 284. Dimensions in meters. Scale, 1:80.

This is a well developed example of heat recuperation by the counter current system. It is true recuperation inasmuch as the heat is recovered from the waste products of combustion and is not abstracted from an active part of the furnace. It may be remarked in passing that the accompanying engraving shows the lowest row of retorts resting directly on the hearth

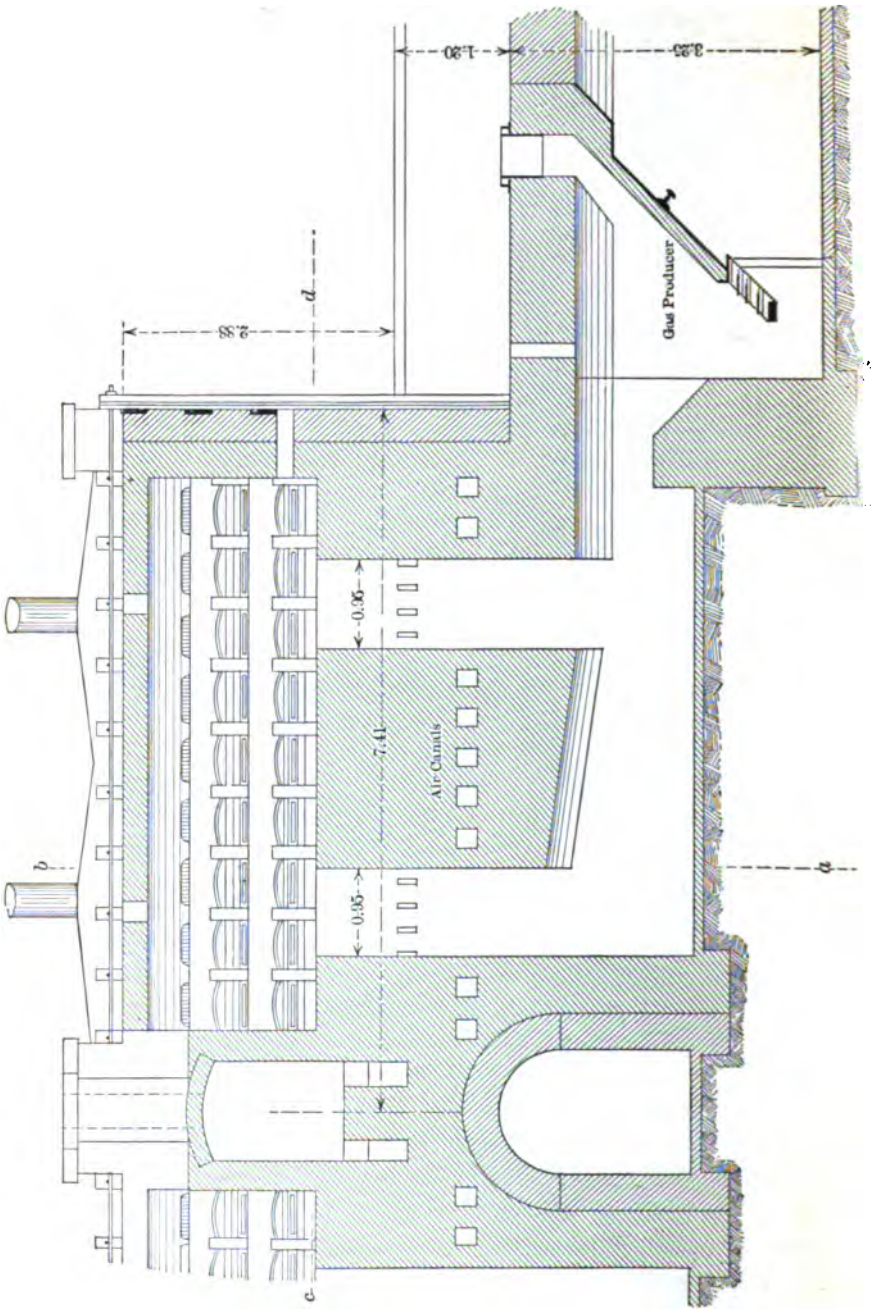


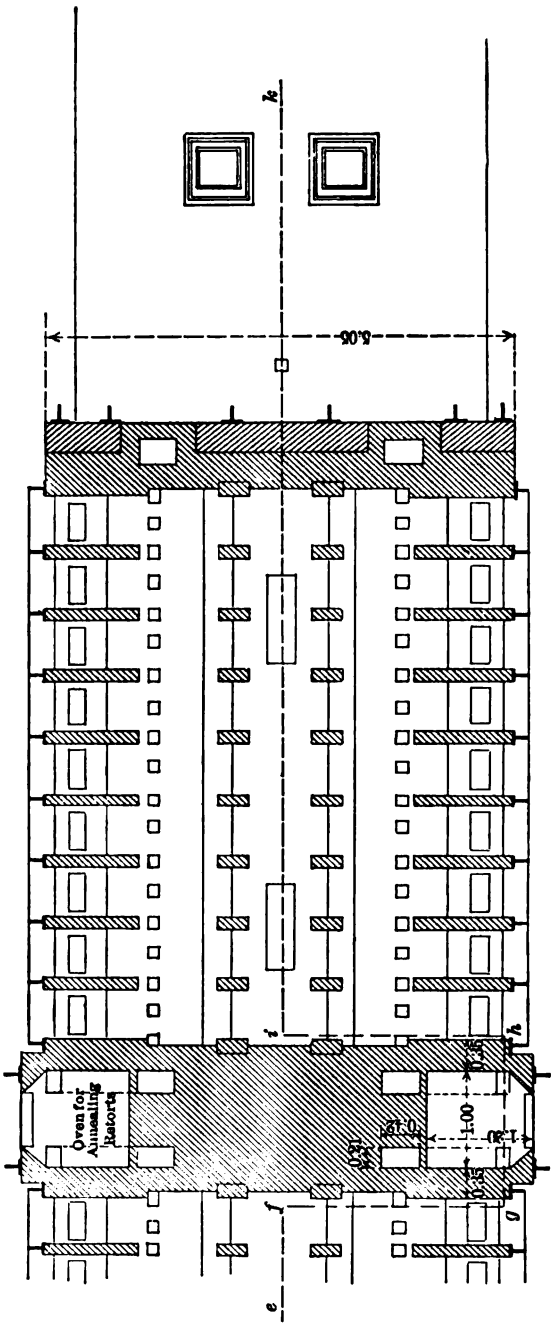
FIG. 284. GAS-FIRED RHENISH FURNACE.

of the combustion chamber, as was the practice with the Rhenish furnaces of that date.

A furnace similar to those which are employed at Valentin-Cocq, but with gas firing and a certain recuperation of heat, is illustrated in Figs. 283, 284 and 285. Two furnaces, each comprising 108 retorts, arranged in three rows of 18 each per side, are grouped in a massive. At each end of the massive there is a gas producer, with sloping breast and step grate, which delivers its gas into a canal extending longitudinally under the hearth. The gas from that canal rises into the combustion chamber through two shafts. The air for secondary combustion is delivered from a main canal at one side of the massive, below the floor of the furnace house, into channels under the flues which carry off the products of combustion. These channels communicate with the gas shafts. The products of combustion escape through openings in the hearth of the furnace into flues which lead to chimneys at the ends of the furnace. Between the two furnaces of the massive there are two ovens for annealing retorts. Two hoods on each working side of the furnace carry off the fumes. The niches are divided by partitions of fireclay tiling and shelves, the latter having iron extensions in which there are holes to drop the retort residues through. The residues collect in the pockets under the lowest shelves.

Siemens System.—The furnaces of the Rhenish type most recently constructed in Belgium incorporate the Siemens system of heat regeneration.

At Overpelt the Siemens furnaces have 216 retorts, arranged in three rows of 36 each per side. The massive is divided by interior walls into four compartments. Two massives are contained in a building 15·4×50 m., affording a space 5·2 m. between the faces of the furnaces and the longitudinal walls of the building. Each massive is supplied with gas from its own generator, situated outside of the building, but the products of combustion of two massives escape through a common chimney, 40 m. high, which is divided into two flues. The retorts are elliptical, 0·17×0·30 m. inside and 1·60 m. long. The open end of the retort beneath the condenser is closed by a removable plate and the retort is charged and discharged without removing the condenser, as in the Silesian furnaces. The niches are divided horizontally by cast iron plates with holes for the descent of ashes and can be closed in front by means of sheet iron doors. Directly above the niches of each side there is a longitudinal flue built with the furnace, communicating at each end with a sheet iron chimney 0·5 m. in diameter and 8·5 m. high, which serves to carry off the dust raised in removing the residues from the retorts; it will be observed that there are four of these ventilating chimneys per massive. Projecting 0·85 m. from each face of the furnace, above


Oven for Annealing Retorts
1.00
0.30
0.35
0.40
0.50
0.60
0.70
0.80
0.90
1.00
1.10
1.20
1.30
1.40
1.50
1.60
1.70
1.80
1.90
2.00
2.10
2.20
2.30
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2.80
2.90
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3.10
3.20
3.30
3.40
3.50
3.60
3.70
3.80
3.90
4.00
4.10
4.20
4.30

the openings of the niches, there are four hoods of which each connects with a chimney 0·65 m. in diameter and 8 m. high to remove the gases issuing from the prolongs during the distillation; there are therefore eight of these hoods with their respective chimneys per massive. The floor of the furnace house is 2·5 m. above the ground level. The works at Overpelt comprise six furnaces, or massives, of the type described above; their production of spelter in 1898 was 4760 metric tons.[1]

The Société Anonyme de la Nouvelle Montagne at Engis had at the beginning of 1899 an installation of four Siemens furnaces and has since built more of them to replace some of the old Liège furnaces. The Siemens furnaces of those works have 240 retorts, arranged in three rows of 40 each per side. As at Overpelt the furnace is subdivided into four compartments. The retorts are of 0·18×0·26 m. inside dimensions and vary in length between 1·25 and 1·65 m. The regenerative chambers are built beneath the combustion chambers, as at Overpelt, and there is a similar arrangement for carrying off the dust during removal of the residues. Two furnaces are built in a house, of which one is 57 m. long, affording a space of 13 m. between the two furnaces, and the other is only 44 m. long, affording a space of only 4·4 m. between the two furnaces. The inside width of the houses is 13·75 m. and 16·9 m. respectively, the space in front of the furnace faces being 4·4 m. and 6 m. respectively. The working floors are 2·4 m. above the ground level. The gas producers are situated outside of the furnace houses. There is one for each furnace. All of them have grates and are water-sealed. Two of them are blown with steam and air by means of Körting injectors; the other two with air alone by means of a fan blower.[2]

The newer furnaces at Prayon are similar in design and arrangement to those at Engis. The retorts are 0·18×0·255 m. inside and 1·25 to 1·60 m. long. There are 240 per furnace, arranged in three rows of 120 each per side. A kiln for annealing retorts is built between every pair of massives.

Belgian Furnaces.

The Belgian, or as it is sometimes called the Liège type of furnace experienced no radical changes from Dony's original design until the introduction of gas firing. There were numerous improvements, however, in the details of construction tending to increase the strength and durability; and the dimensions of the combustion chamber and the number of the retorts were gradually enlarged. The size of the chamber and number of

[1] The general proportions of the furnaces at Overpelt, and their arrangement, together with a new system of ventilation of the furnace house, are illustrated in the sketches on pp. 629 and 630.

[2] Firket, loc. cit.

retorts varied greatly in the old furnaces according to the character of the coal available for heating them. The length was always limited by the physical inability of the stoker to throw the coal properly over the grate beyond a certain distance; the width was similarly limited by the inability of the chargers to fill the retorts exceeding a certain length and the inability of the loaded retorts to support themselves; while the height was governed by the distance above the grate to which the requisite temperature could be maintained and this is to a considerable extent determined by the character of the coal. With long flaming fat coal the combustion chamber could be made higher than with short flaming lean coal and more rows of retorts could be used. The employment of clinker-grate firing lessened the disadvantage of lean coal and with the introduction of gas firing the character of the coal became of still less importance and the height of the furnace as a factor of its size was eliminated entirely.[1]

In the design of the earlier furnaces the tendency was to increase the height of the combustion chamber and the number of rows of retorts. This was conducive to economy of fuel but was unfavorable to recovery of metal, because the retorts of the upper rows could never be heated properly without butchering those of the lower rows, while if the latter were not overheated the former would fail to give a good extraction of metal; moreover these very high furnaces were always trying on the men who were obliged to stand on a table in order to perform the duties of the maneuver for all the rows above the fifth. The modern tendency, both in Belgium and the United States, is strongly toward reducing the height of the furnaces, which thanks to gas firing can be done without sacrificing anything except floor space. The Société Anonyme de la Vieille Montagne at its works at Angleur, Belgium, first reduced the height of its furnaces to five rows of retorts; in its latest constructions this has been further reduced to four rows. The trend of zinc smelting practice in Belgium toward the adoption of the Siemens furnace with only three rows of retorts has already been referred to in a previous section of this treatise.

Belgian furnaces may be classified as direct fired and gas fired, but there is no sharp division between these systems, which merge into one another by difficultly definable gradations. It is easy to make the distinction between furnaces which are heated by grates directly under the retorts on the one hand and those which are heated by gas derived from a source entirely apart from the furnace on the other hand; but between these extremes there are furnaces commonly known as gas fired (rarely as semi-gas fired) in which the producer is an integral part of the furnace; and

[1] This statement will be explained in the discussion of the Hegeler and Iola furnaces.

direct fired furnaces operated with deep beds of fuel on the grate, which effect an important gasification of the coal in its combustion. With this explanation it will suffice to draw a line midway between the extremes, which will correspond with the nomenclature of practice, although perhaps unscientific. Each class of furnace may be subdivided into recuperative and non-recuperative, but heat recuperation in connection with direct firing is very rare.

Direct Fired.

EARLY LIÈGE FURNACES.—The first Liège furnaces were short and high, comprising 46 small cylindrical retorts arranged in eight rows.[1] There was only one distillation chamber and one brigade of retorts, i.e., only one working face. The roof arch was sprung from end to end of the furnace, wherefore the highest row of retorts comprised a smaller number than the lower, for which reason the total number was not a multiple of the number of rows.

Furnace used at Moresnet.—A furnace with 61 retorts, of the original Liège type, which was used at Moresnet in the '50's, is illustrated in the accompanying engravings.[2] These furnaces were built in pairs, end to end, communicating with a common chimney. The combustion chamber of each furnace was 2·45 m. long, 1·5 m. wide and 3·2 m. high. It contained 61 retorts arranged in seven rows of eight and one row (uppermost) of five. There was also a row of eight larger tubes, or cannons, immediately over the grate, by which the temperature of the furnace was regulated. These cannons being placed directly under each tier of retorts protected the latter from sudden variations in the temperature of the fire and turned the flames up between them.[3] The cannons were also penetrated by holes through which air was admitted into the combustion chamber and by regulating the supply of air the temperature could be controlled. The same thing is accomplished by admitting air into the furnace through holes punched in the luting of the front wall between the retorts.

The retorts of the furnace at Moresnet were 1·1 m. long and 0·15 m. in diameter; they were set at inclinations of 0·078 to 0·158 m. according to the purity of the ore. At the rear end they were supported on shelves

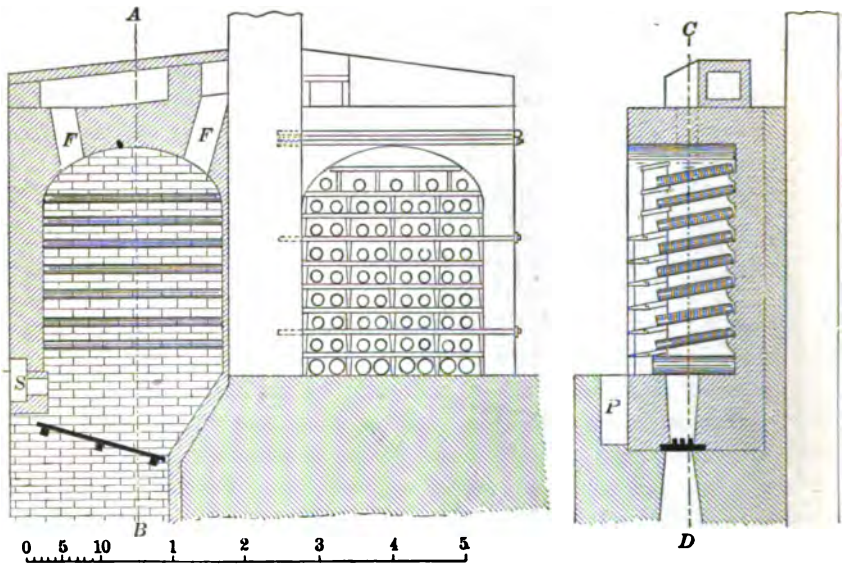
[1] De Sinçay, The Mineral Industry, VIII, 654.

[2] Berg- u. Hüttenm. Ztg., 1859, p. 405; ibid. 1860, p. 3.

[3] Instead of using cannons the retorts were protected in some of the early furnaces by interposing an arch pierced by numerous holes between the lowest row and the fireplace. These devices ceased to be of use after the introduction of clinker-grate firing, whereby jets of flame or of relatively cold air, which might strike the retorts, were no longer produced.

built in the back wall, just as in the furnaces of the present time; at the front end those of the six lower rows rested on fire clay tiles, which were supported longitudinally by iron plates as at present and at the ends by fire brick pillars, but the two upper rows rested only on iron plates. The plates and pillars divided the front of the furnace into niches, each of which contained two retorts, except the highest row wherein there was only one to a niche.

The fire box was 0·4 m. wide at the grate and 0·45 m. at the top. The grate had such an inclination that it was 0·8 below the hearth of the

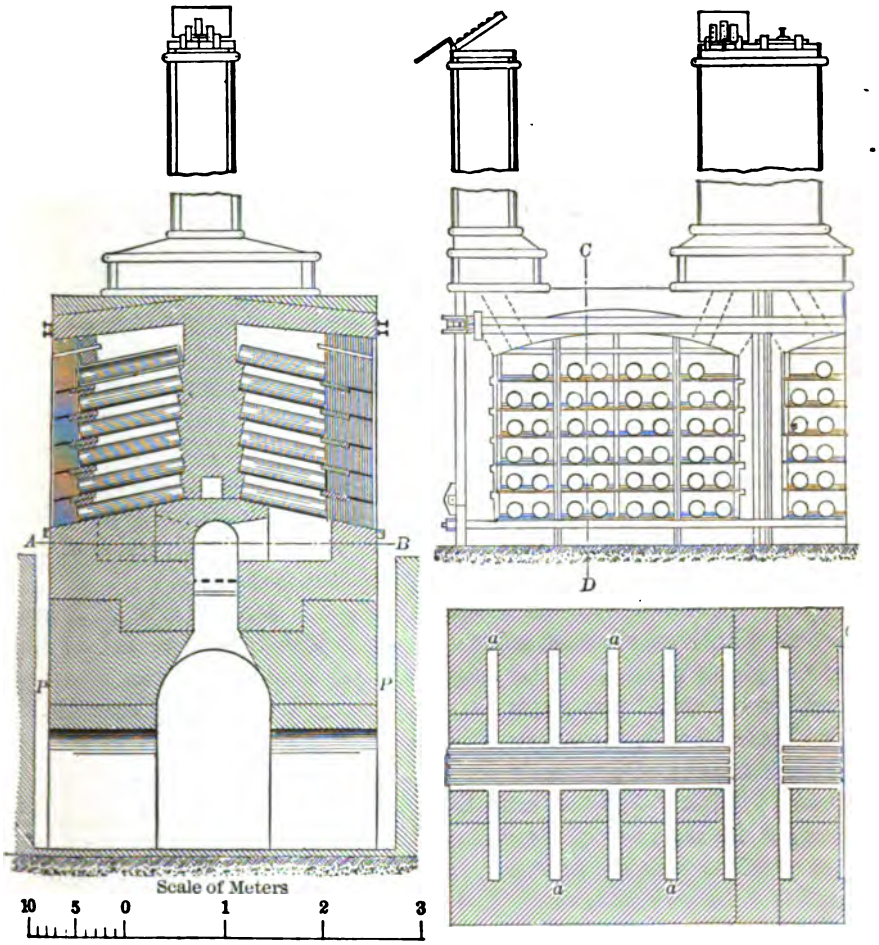


FIGS. 286 AND 287. FURNACE USED AT MORESNET.

Fig. 286: Section on line CD of Fig. 287 and front elevation. Fig. 287: Section on line AB of Fig. 286.

combustion chamber at the outer end and 1·1 m. at the inner. There were three grate bars. The products of combustion escaped through the flues *FF* into the main flue which communicated with a chimney 4·5 m. high, or else with a calamine calcination chamber and thence to the chimney. The condensers were fitted with sheet iron ballons of the form shown in Fig. 249, which were "an improvement on the simple sheet iron cones that had been used formerly at Engis" (Kerl). In Fig. 287, *P* is a pocket for collecting the retort residues. I have described at such length this Liège furnace of 40 years ago to show how little has been the change in this type.

Combine two furnaces back to back as a pair and two pairs end to end as a massive and save for a few structural details we have the direct fired Belgian furnace of the present day. In fact some of the older furnaces at Bley-



FIGS. 288 TO 290. NOUVELLE MONTAGNE FURNACE.

Fig. 288: Transverse vertical section, the left hand side being on line CD of Fig. 289 and the right hand side on a line passing through the flame ports *aa* of Fig. 290. Fig. 289: Front elevation. Fig. 290: Horizontal section on line AB of Fig. 288.

berg, Boom and Seilles, in Belgium, are even now substantially of the above described type.

Nouvelle Montagne Furnace.—A departure from the typical Liège fur-

nace in so far as the arrangement of the fireplace is concerned was made by the Société de la Nouvelle Montagne at its works at Engis. This was a double furnace, of which two were joined end to end to form a massive. Each double furnace had a long fireplace in the middle, parallel with the long axis of the furnace, and a deep ash pit under the grate. The middle wall dividing the combustion chamber was supported on an arch over the fireplace; just over the arch it was pierced by a longitudinal cooling flue. On each side of the fireplace there were five ports through which the flames passed into the combustion chambers and thence escaped through ports to a chimney (7 m. high) standing on each end wall of the furnace, i.e, there were three chimneys to a massive, one at each extremity and one large chimney with two flues on the middle wall separating the two furnaces of the massive. Each furnace had two brigades of retorts, arranged in six rows, with eight in a row, except in the highest where there were only six; consequently there were 92 retorts per furnace, and 184 per massive.[1] These furnaces, which were in use in 1871, apparently have given satisfactory results (vide p. 535) inasmuch as 13 of them were still in use at Engis at the end of 1898.[2] At that time the furnaces had 90 to 106 retorts (180 to 212 per massive) arranged in five or six horizontal rows. Some had cylindrical retorts 0·17 m. in diameter inside; others elliptical retorts, 0·175× 0·195 m. In length each kind was 1·25 to 1·30 m. Recently many of these furnaces at Engis have been replaced by Siemens furnaces.

Borgnet Furnace.—Alfred Borgnet designed a peculiar type of Belgian furnace with a single combustion chamber, which instead of having long narrow grates parallel to the longitudinal axis of the furnace and fired from the ends, had five grates set at right angles to the long axis and inclined parallel with the retorts which were fired from the back of the furnace. Furnaces of Borgnet's design were erected at the Morriston zinc works of Vivian & Sons, near Swansea, Wales. They had 120 retorts arranged in six rows of 20 each. The retorts of the lower four rows were provided with air flues, such as are shown in Fig. 169. Each grate was 24×7 in. in dimensions. The flames escaped through 17 holes in the arch of the combustion chamber into a main flue extending longitudinally over the furnace and thence to a chimney 23 ft. high. The arch of the furnace was supported on the front side by 11 cast iron pillars, cast in one piece, 4 in. wide over all, the sides being 0·5 in. thick and the space 3 in. wide between them being filled with fire clay tiling. Cast iron plates 20 in. long rested on lugs cast on the pillars, fire brick tiling being laid on them

[1] Massart, Revue Universelle des Mines, 1871, XXIX, 313.
[2] Ad. Firket, Annales des Mines de Belgique, VI, ii.

in the usual manner. The end walls of the furnace were 4 ft. thick and the length of the combustion chamber between them was 20 ft. 6 in. The width of the furnace over all was 7 ft., the height from grate to arch was 14 ft., and height in front from the lowest shelf to the spring of the arch was 10 ft. The distance between the shelves of the first and second rows was 20 in., of the third and fourth 19 in., of the fifth and sixth 17 in. The arch was 10 in. thick. The walls of the furnace were supported by buckstaves of iron rails. This furnace was capable of smelting 2100 kg. of ore assaying 49 to 50% Zn, mixed with 1000 kg. of reduction coal per 24 hours, yielding 785 to 850 kg. of zinc. The consumption of heating coal amounted to two tons per ton of ore, and of retorts 1·6 per ton of ore. The furnace crew consisted of three men per shift.[1]

American Belgian Furnaces.—The direct fired Belgian furnaces employed in the United States conform essentially to the original Liège design, but present minor differences because of adaptation to local conditions. This is especially the case between the furnaces of New Jersey and Pennsylvania, which burn anthracite coal, and those of the Western States, which burn bituminous.

Western Type.—The typical, direct fired Belgian furnace which is used in the Western States is illustrated in the accompanying engravings. There is no essential deviation from that type. The deep fireplace and clinker grate, high, open ash pit, combination buckstaves and stanchions, stiffening the furnace laterally and supporting the skewback of the arch, are universal; so too is the absence of end buckstaves and the stiffening of the furnace longitudinally by means of rods carried horizontally around it.[2] Furnaces of this type were built in the first works in Kansas and they were imitated until finally displaced by the natural gas furnaces.

The retorts are invariably cylindrical, 8 in. in diameter inside and 10·5 to 11 in. outside, and are generally 48 in. long outside. Sometimes the two lower rows are only 46 in. long, the upper being 48 in., and occasionally they

[1] Georges Borgnet, Revue Universelle des Mines, November and December, 1877; Berg- u. Hüttenm. Ztg., 1878, p. 387. This paper describes with much detail the Welsh practice in zinc smelting at that time.

[2] This construction is similar to that of some of the old direct fired New Jersey furnaces, which had their angles encased by heavy castings provided with lugs through which rods extending laterally and longitudinally were bolted. There was a longitudinal rod along the shelves supporting the fourth row of retorts (counting from bottom upward) and a lateral rod on the same level at each end; also a lateral rod just above the level of the lowest row of retorts; and a lateral and longitudinal on the level of the skewback of the arch; the longitudinals last mentioned were trussed with the beam which supported the skewback and took the place of buckstaves. The strongest construction is to provide vertical buckstaves for the ends of the furnace and unite them by means of tie rods above the roof in the usual manner of supporting metallurgical furnaces. An example of this method may be seen in Figs. 296 to 298.

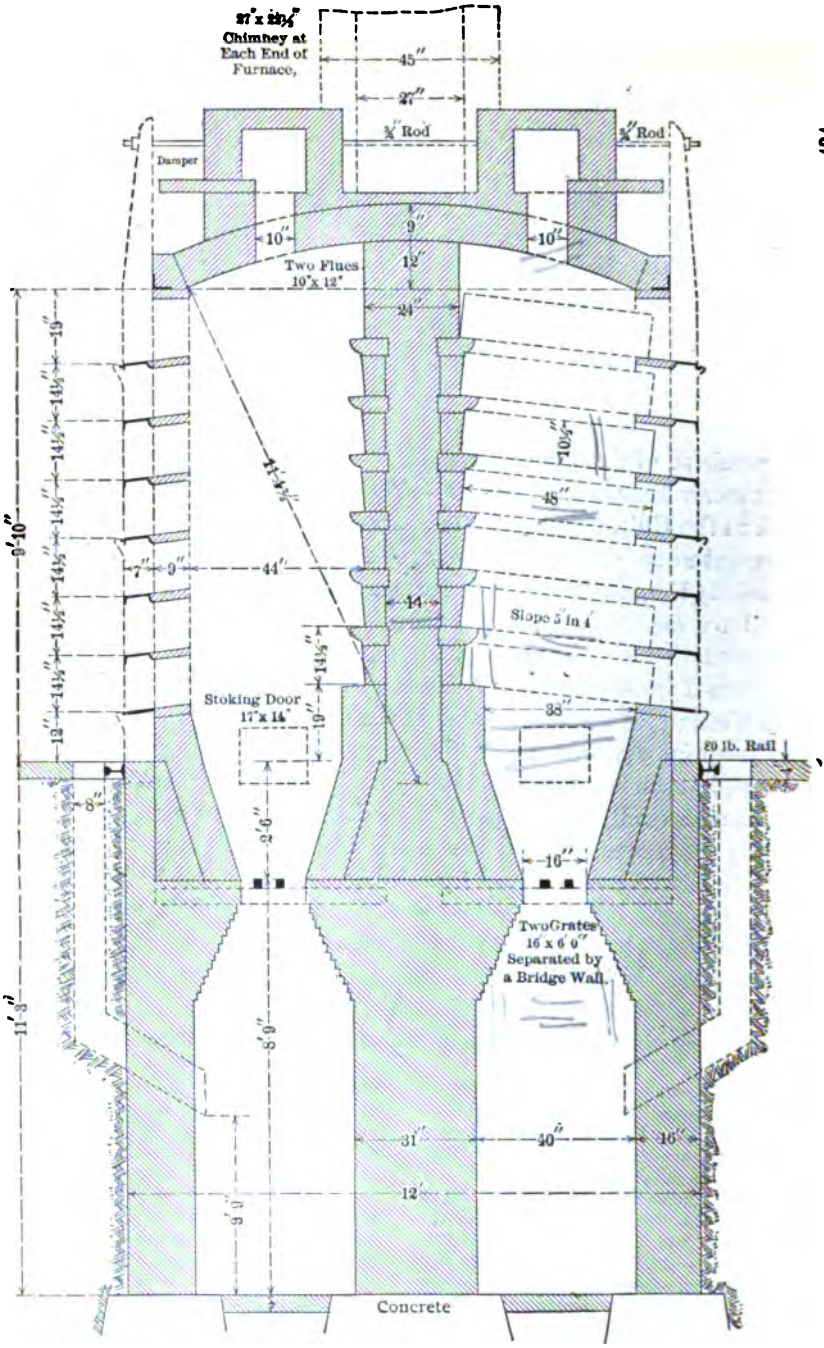


FIG. 291. AMERICAN BELGIAN FURNACE (WESTERN TYPE).
Scale, ¼ in. = 1 ft.

FIG. 292. BELGIAN DISTILLATION FURNACE, RETORTS REMOVED.

are 48 and 50 in. respectively. By far the most part of the furnaces have 224 retorts arranged in seven rows of 16 each, i.e., 112 per side. Less commonly there are furnaces with $2\times7\times14=196$ and $2\times8\times16=256$. Each combustion chamber has two grates, separated by a transverse wall in the middle, the wall rising a few inches above the bed of coal which is to be carried. The grates for the 224 retorts furnace are approximately 6·5 ft. × 15 in.=8·125 sq. ft., or 32·5 sq. ft. for the four. Variations are 7×14 in.=8·167 sq. ft. and 6 ft. × 16 in.=8 sq. ft. The grate is set from 8 to 9 ft. above the bottom of the ash pit. It is composed of two wrought iron bars, 2 in. square. The coal is so pyritous that the bars are rapidly corroded, the average life being about three months. At Nevada, Mo., wrought iron 2 in. pipes, through which water circulated, were used. It was said that they were cheaper than solid bars, lasted longer and were more easily cleaned of clinker.

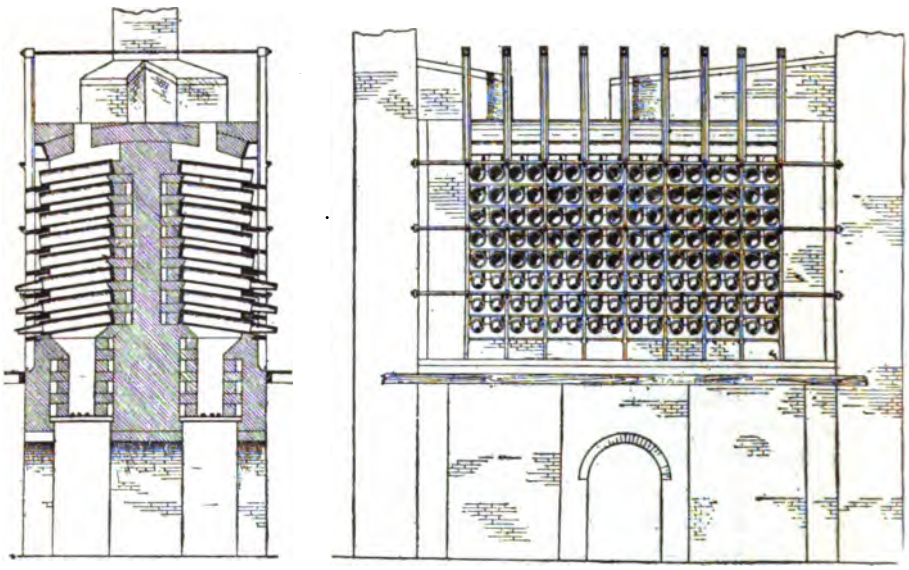
The column of coal above the grate bars is about 3 ft. high. Its surface is larger than that of the grate because all the sides of the fire box slope towards the grate,[1] and it is maintained from 12 to 15 in. below the lowest part of the lowest retorts, the top of the fire door being about 12 in. below the top of the lowest front shelves. There is a fire door at each end of each combustion chamber. The fire doors are about 17 in. wide and 14 in. high. They are framed with a fire clay tile for the sill, two fire clay posts, and a tile 9 in. thick as cap. In the brickwork over the cap a T rail is set horizontally to relieve the cap from the weight of the wall. Outside of the door there are two iron brackets, one on each side, on which an iron plate can be set, sloping down to the sill of the door. After stoking, coal (slack) is heaped on this plate until the door is sealed. When the fire has to be stoked the coal seal is pushed inside and the door is thus opened for fresh coal.

In the roof arch of each combustion chamber there are two flues, usually 10×12 in., of which the centers are about 40 in. from the end walls. Formerly these flues were made smaller and there was a third flue between them, but the latter was abandoned as unnecessary. At each end of the furnace there is a chimney about 45 ft. high above the grate. They rest partly on the end wall and partly on a pier built up from the ground. They are made from 36×45 in. to 45×45 in. outside, and 18×27 in. to 27×27 in. inside dimensions. They are bound together by angle bars encasing the corners and horizontal engirdling rods; the two chimneys are usually connected by a rod or chain to restrain them from falling outward. They are set on the furnace sometimes parallel with the walls of the latter and sometimes diagonally; the

[1] If not built originally with that shape, it soon acquires it naturally.

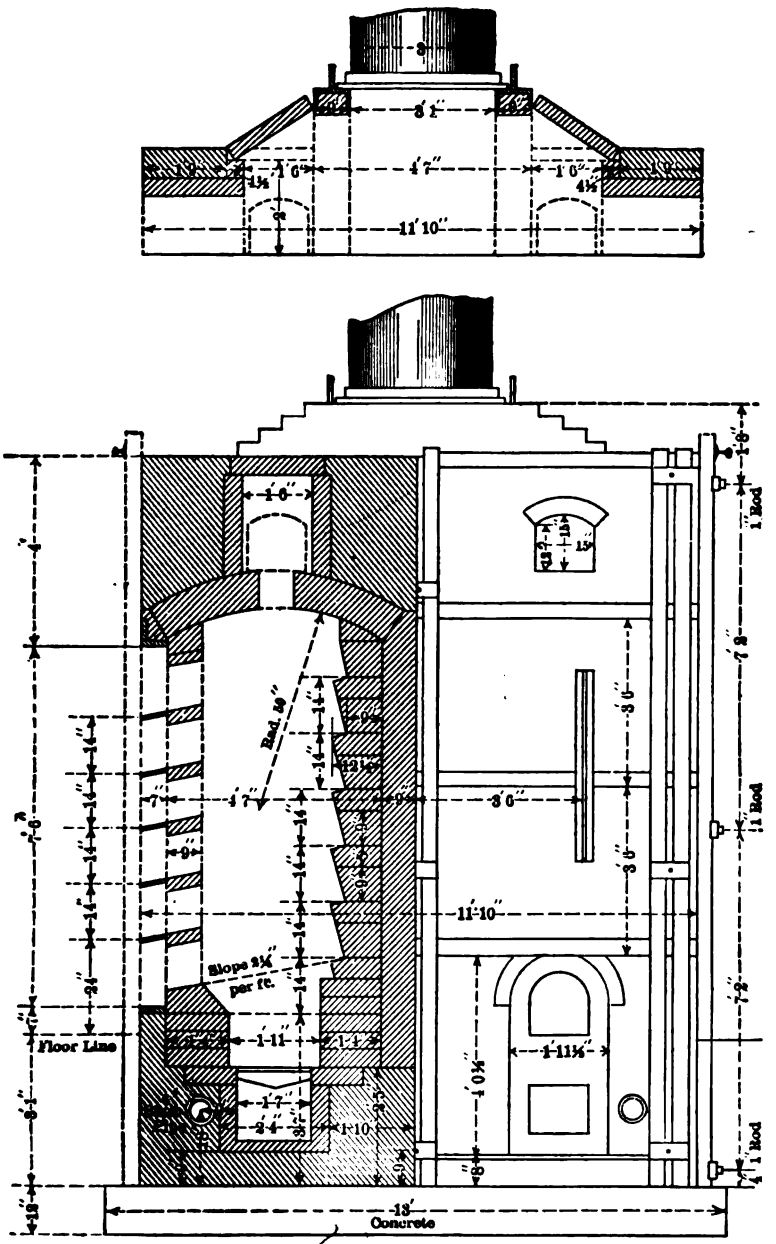
latter method permits the canals connecting with the flues in the furnace to be shorter and more direct; the former method necessitates that the canals be curved.

The corners of the furnace are encased by 4×4×0·5 in. angle bars. The skewback of the arch is supported on a 5·5×3·5×0·5 in. angle bar. The buckstaves are tied together by rods of 0·75 in. diameter. In the floor of the furnace house along each face of the furnace there is a rectangular opening into iron hoppers, two on each side, of which the chutes lead into the ash tunnel. These hoppers are installed large enough to hold the whole



FIGS. 293 AND 294. BELGIAN FURNACE USED AT COLLINSVILLE, ILL.
Fig. 293: Transverse vertical section. Fig. 294: Front elevation.
These furnaces are worked from an elevated floor.

drawing of retort residues after each charge. Generally each furnace is contained in its own house, which is a rude frame shed with a high, steeply sloping, pyramidal roof, left uncovered over the furnace. The floor area is approximately 50×50 ft. inside of the joists. In the sides and ends there are large openings, which can be closed by sliding doors to protect the furnace from strong winds. The floor is laid with brick, except at the ends of the furnace, where there are platforms of plank on a level with the grate. This depression gives easy access to the stoking door and also affords room for the day's supply of coal.



FIGS. 295 AND 296. TYPE OF FURNACE USED IN NEW JERSEY.

Fig. 295: Transverse section through flues leading to chimney. Fig. 296: Transverse vertical section of furnace and end elevation.
Scale, ¼ in. = 1 ft.

A Belgian furnace formerly used at Collinsville, Ill., is shown in the accompanying engravings. It had 2×8×16=256 retorts. A noteworthy feature was the air channels in the middle wall and around the fire box. The furnaces lately in use at Collinsville are practically the same, but the air channels have been abandoned. They have 256 retorts and four grates each 7·5 ft. × 15 in.=9·375 sq. ft., i.e., a total grate area of 37·5 sq. ft. The bed of fuel carried on the grate is 4 to 4·5 ft. thick.

Eastern Type.—The direct fired distillation furnaces used at the works at Newark and Jersey City, N. J., and South Bethlehem, Pa. (and formerly at Bergenpoint, N. J., and Friedensville, Pa.), differ from the ordinary type of Belgian furnace only in the design of their fireplaces, which are adapted to the combustion of anthracite coal from nut to No. 2 buckwheat size. Instead of the deep fireplace and high open tunnel under the grate there is a shallow fireplace and an ash pit only 16 to 18 in. deep under the grate. The ash pit is closed with a tightly luted door and air under pressure of about 4 in. of water (2·3 oz.) is blown in under the grate. The grate is composed of cast iron bars 1⅜ in. thick, which are set closely together and are perforated by conical holes, which are of 0·4 in. diameter on the upper side of the plate and 1 in. on the lower side. This prevents pieces of clinker from being wedged in the holes; anything which gets in them must drop through. There are 100 holes per square foot; consequently the openings in 1 sq. ft. amount to only 12·57 sq. in., or approximately 9%; but the openings between the bars afford a further free space. At the far end of the fireplace there is a fire brick platform level with the grate, or sloping upward from it. In cleaning the fire the live coals from the top are pushed back on that platform. The cinders are then barred up from the grate and scraped out through the fire door. The live coals are then drawn forward over the clean bars, fresh coal is thrown in, the blast is turned on and in a short time the furnace is brought to the required temperature, which is regulated by varying the blast of air. In some of the furnaces a row of cannons under the lowest row of retorts protects the latter from the intense radiated heat of the fire; in other works they have been dispensed with.

The furnaces at Jersey City, Newark and South Bethlehem differ in minor particulars, and the practice at all of the works has changed considerably in the last 20 years. Some of the older furnaces are divided into four sections by a middle transverse parting wall. The Belgian furnace used in the West has also four grates, but the transverse walls are not carried up above the fireplaces. In the early '80's the New Jersey-Pennsylvania furnaces had 112 retorts per side,[1] arranged in seven rows of 16 each. The

[1] F. L. Clerc, Mineral Resources of the United States, 1882, p. 373.

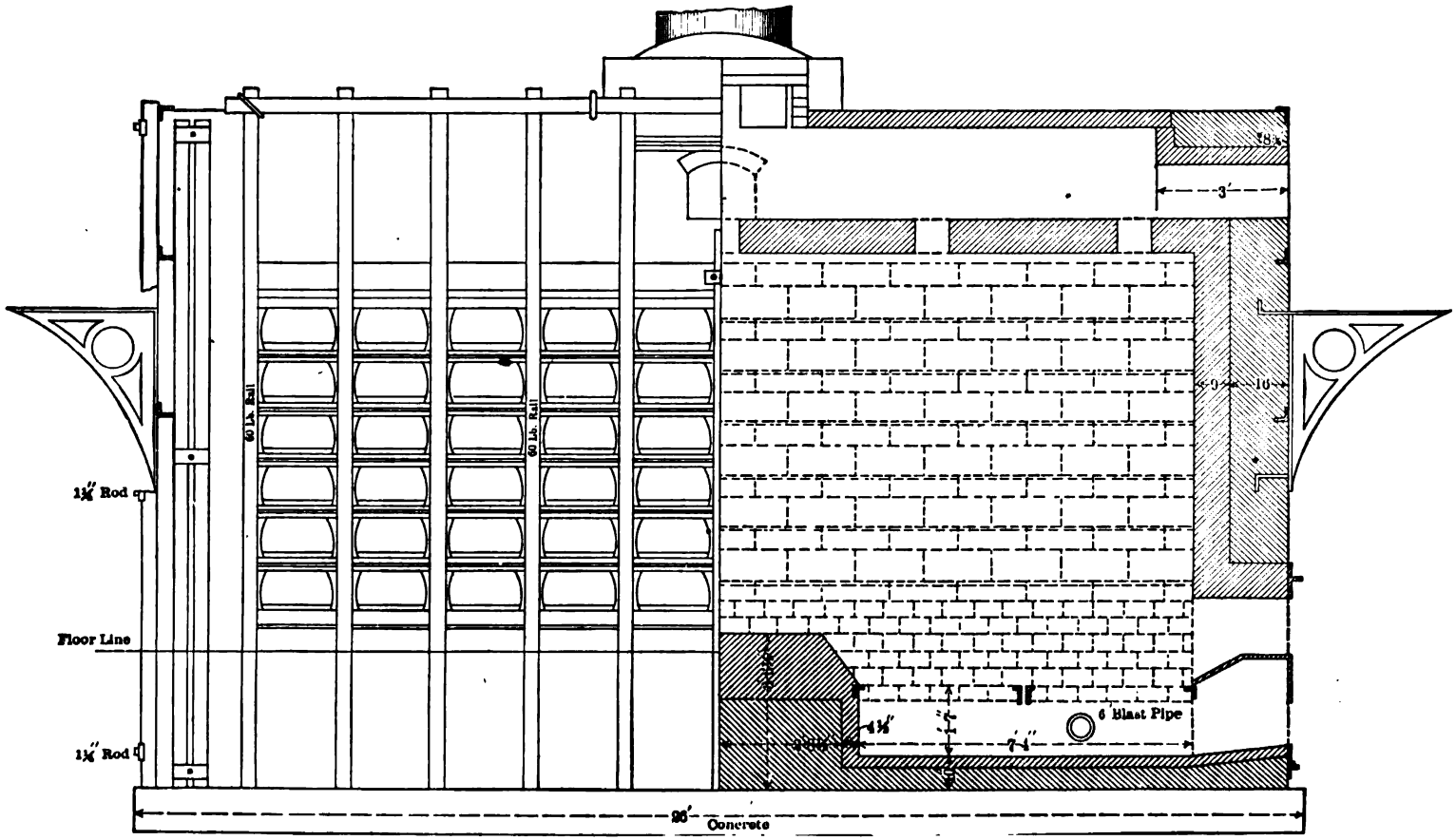


FIG. 297. TYPE OF FURNACE USED IN NEW JERSEY.

Front elevation and longitudinal section.
Scale, ¼ in. 1 ft., approximately.

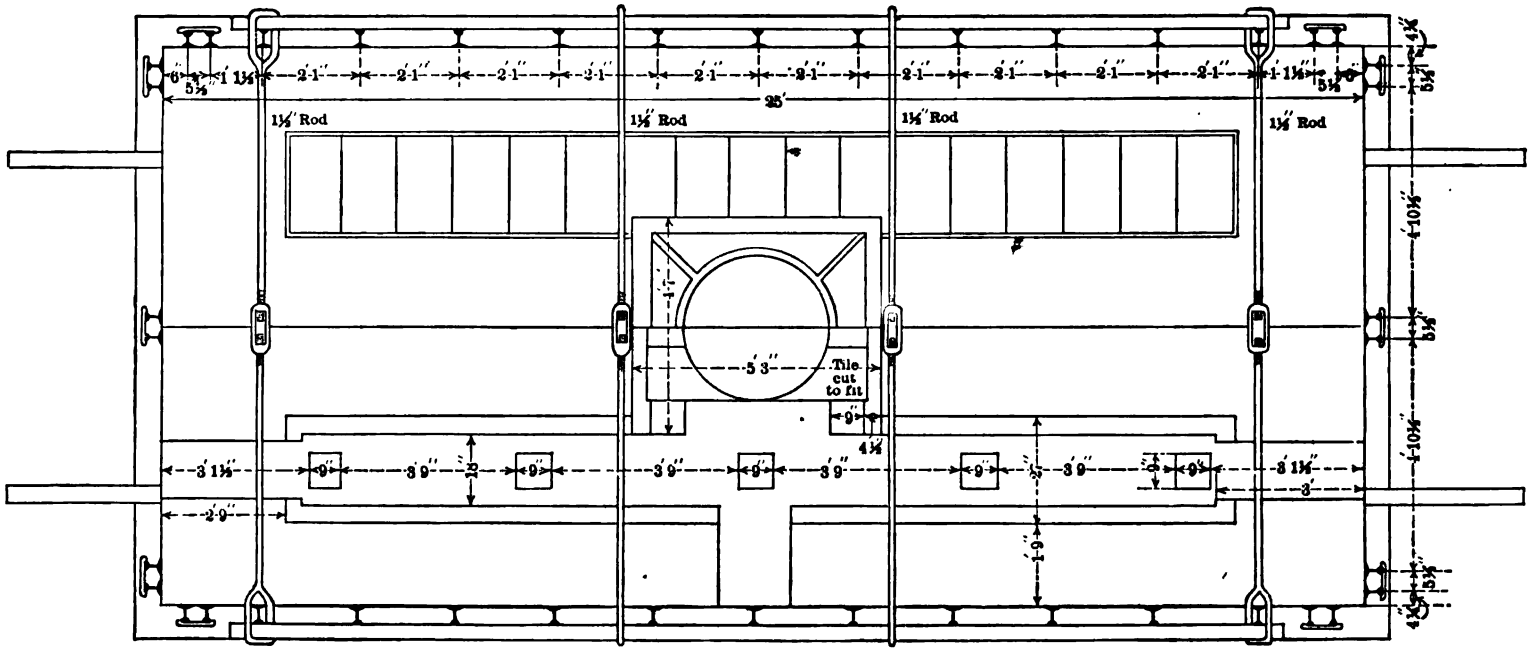


FIG. 298. TYPE OF FURNACE USED IN NEW JERSEY.

Plan.

Scale, ¼ in. = 1 ft., approximately.

furnace was charged twice in 24 hours and the retorts (cylindrical) were of only 6·5 or 7 in. diameter. Later the furnaces were increased in size, charging was done only once in 24 hours, and elliptical retorts were introduced. At Jersey City in 1894 there was one furnace with $2\times6\times18=216$ retorts, but apparently the reduction in the number of rows did not give sufficiently better results to compensate for the reduction in capacity of the furnace, since one of later construction had $2\times7\times18=252$ retorts. The four lower rows of retorts were elliptical, 7×9 in. and 54 in. long; the upper rows were cylindrical, 7 in. diameter and 54 in. long. The longitudinal middle wall of these furnaces was hollow and air was blown in through the flue thus formed. There were no cannons. Each massive had three chimneys, one at each end and one in the center. Prolongs were used on the condensers.

At South Bethlehem, Pa., in the same year there were furnaces with $2\times4\times20=160$ and $2\times7\times20=280$ retorts. Previously all had had 280 retorts, but some had been cut down on the ground that with the seven high furnaces the lower rows had to be butchered to heat properly the upper rows. Cannons were installed beneath the lowest row of retorts. The retorts were of 7 in. diameter and 50 in. length. Prolongs were used on the condensers.

The furnaces used at Friedensville had six rows of 8 in. retorts, 108 per side, 216 per massive. The combustion chamber was divided into four sections. Cannons were used.

One of the latest direct fired furnaces erected in New Jersey (date 1895) is shown in the accompanying engravings. With the consolidation of the zinc mining and smelting business of New Jersey and Pennsylvania in the hands of the New Jersey Zinc Co. the direct fired furnaces of the old works will probably be abandoned and the production of spelter will be concentrated at the new works at Palmerton, where gas firing is employed. Experiments in gas firing were first made with a Rhenish furnace at South Bethlehem in 1894.

Furnaces at Pulaski, Va.—A form of Belgian furnace, used at the works of the Bertha Mineral Co., at Pulaski, Va., is shown in the accompanying engraving. These furnaces have single combustion chambers with 140 retorts, arranged in seven rows of 20 each. The retorts are of elliptical cross section, 8×10 in. inside and 48 in. long.

OTHER FURNACES.—As has been previously remarked, the combustion of coal in a deep bed on the grate involves the general principle of gas firing. The coal is largely converted into gas which burns around the retorts in the large combustion chamber and to a considerable extent escapes unconsumed

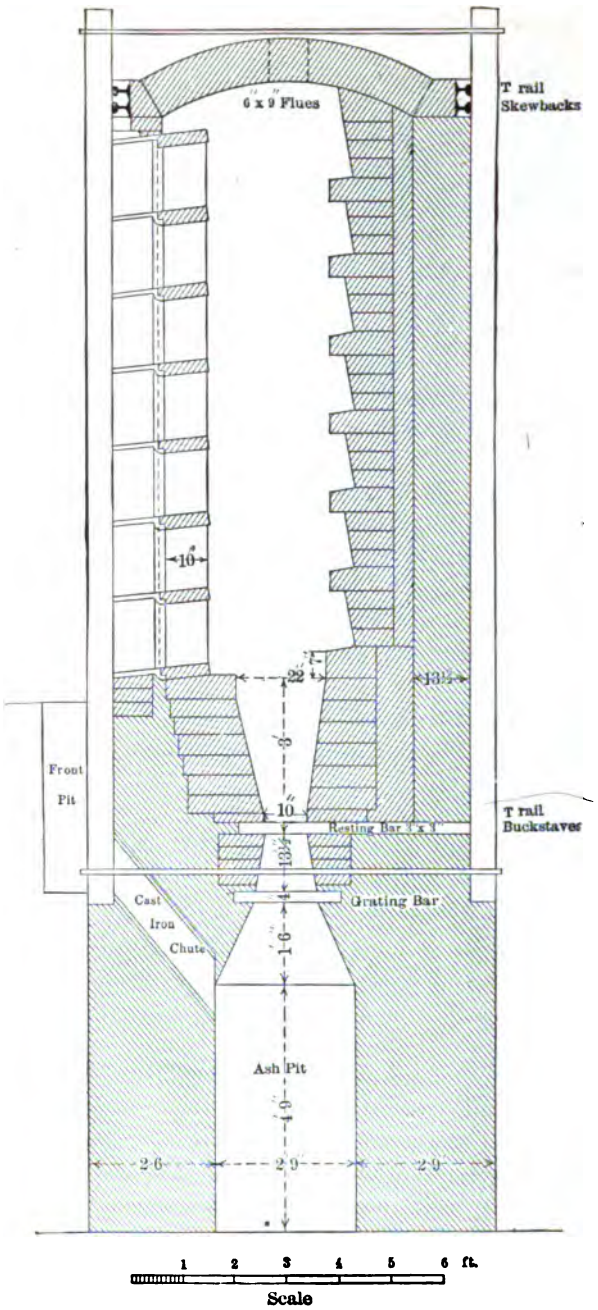
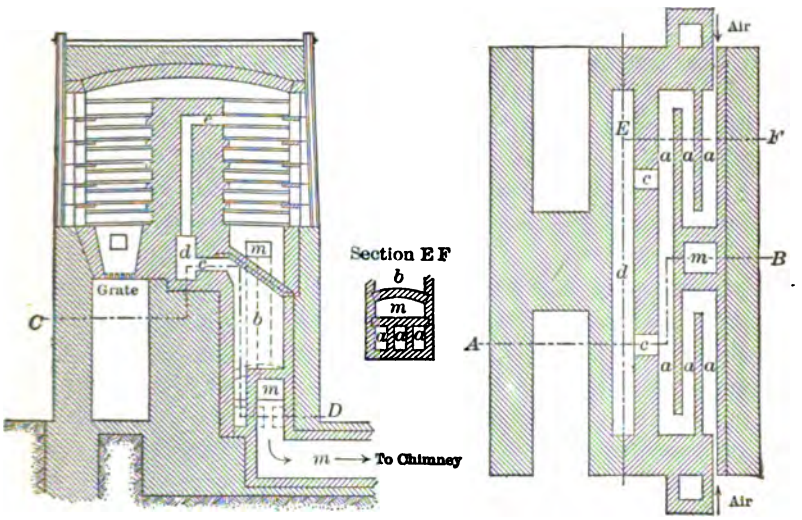


FIG. 299. FURNACE USED AT PULASKI, VA.

Transverse vertical section.

therefrom as is manifested by the flames issuing from the chimneys when the stoking is being pressed. Several furnaces have been designed to minimize this great waste of heat by effecting more complete combustion of the gas, combined possibly with a partial recuperation of heat.

Hauzeur Furnace.—Hauzeur developed logically the idea of reducing the waste of heat and his furnaces have been used successfully in Belgium and Spain.[1] He designed a double furnace in which the middle wall, built thicker than ordinarily, was stopped short of the roof arch, so that there was free communication between the two combustion chambers. Only one



FIGS. 300 TO 302. HAUZEUR FURNACE.

Fig. 300: Transverse vertical section on line AB of Fig. 302. Fig. 301: Section through flues on line EF of Fig. 302. Fig. 302: Horizontal section on line CD of Fig. 300.

of the latter was provided with a fireplace. The burning gas from that fireplace rose between the retorts, drew over the middle wall into the other combustion chamber, passed down between its retorts and escaped through the flue *m* to the chimney. In the part of the furnace lying under the second combustion chamber the products of combustion were made to pass over a labyrinthine air canal *aaa*. Air admitted through the latter passed into the chamber *b* and thence through *c*, *d* and *e* into the second combustion chamber, where it furnished oxygen to the unburned gas coming over from the first chamber. The furnace was a truly recuperative one, although

[1] German patent, 3,729, Sept. 15, 1877; Spirek, Oest. Zts., 1881, p. 335; Dingler's Polytech. Journ., Vol. CCXXXV, 221.

the system of heat recuperation was not highly developed. It is interesting as affording an example of the application of the counter-current system of heat recuperation to a direct fired furnace; for the Hauzeur furnace can no more be considered a gas fired furnace than can any of the many forms in which a deep bed of fuel is carried on the grate.

Kämmerling Furnace.—Herman Kämmerling, of Girard, Kan., patented a modification of the ordinary Belgian zinc smelting furnace, wherein there are two combustion chambers, separated by a central longitudinal wall, as in the usual type of double furnace, of which only one is provided with a grate. The gases of combustion pass from the fire on the grate between the retorts of that side of the furnace, and then through ports arranged at intervals near the top of the dividing wall, and down between the retorts of the other side of the furnace, escaping through openings in the floor of the latter, thence passing through suitable conduits to the chimney.[1]

The idea of the inventor of this furnace was that the ordinary Belgian furnaces in use in Kansas at that time could be easily and cheaply converted into the new type; and in fact a large number of furnaces at Pittsburg and Weir were thus altered, the middle wall being cut down so as to permit the flames to pass over it just under the roof arch, and a hearth, with suitable openings for the escape of the gas, being placed in the opposite combustion chamber, which communicated with the chimneys by a flue built in the former fireplace. All the old flues in the roof arch were of course closed.

At first these furnaces gave such remarkable results in respect to saving in fuel consumption and economy in labor that they gained the name of "Klondike" furnaces, but it was subsequently found that these economies were realized only in the early part of the campaign, their efficiency falling off with the progress of the latter until the results became inferior to those of the ordinary Belgian furnaces. This was due especially to the formation of obstructions in the flues for the escape of the products of combustion by the slag dripping from the retorts and walls of that side, whereby control of the firing was lost and it became impossible to maintain the proper temperature in the combustion chamber, with the result of a decreased extraction of metal. For this reason the furnaces which had been converted were changed back to their original form.

The experience was interesting and valuable as showing the economy which it is possible to effect by the introduction of gas, or semi-gas firing with the ordinary Kansas coal slack, notwithstanding the failure of the original and more elaborate attempts at Pittsburg, Kan., and Rich Hill,

[1] United States patent, No. 610,540, Sept. 13, 1898.

Mo., to which reference will be made subsequently. There was nothing new in the idea of the Kämmerling furnace, except in so far as it related to the cheap alteration of the old Belgians, which was in principle substantially the same as Hauzeur's. Its lack of ultimate success was due rather to the crudity of the design than to error in principles.

Gas Fired, Non-Recuperative.

In gas fired Belgian furnaces the retorts are heated either by the combustion of producer gas or of natural gas. The latter finds application only in the United States. Although the peculiar characteristics of this powerful fuel necessitate important modifications in the furnace design, the general principles remain the same. With producer gas firing, the gas may be derived either from a producer built integrally with the furnace, or the producer may be situated at some convenient point apart from the furnace, the gas being conveyed to the latter through a canal, which may be arranged in various ways, as will appear in the engravings accompanying this section.

The application of gas firing, at first producer gas firing only, permitted Belgian furnaces to be built larger, affording at the same time the advantages derived from a more equable temperature and a plenum of pressure in the combustion chamber, together with economy in fuel. For these reasons the principle has been now extensively adopted in Belgium, although numerous examples of the older type still survive. Save in connection with the more modern furnaces, however, the producers which are employed are chiefly of the Boetius type, forming an integral part of the furnace, and with that system not all kinds of coal are suitable, wherefore there are important smelters in Belgium who still adhere to the old method of direct firing. Touching upon that point, M. de Sinçay, director general of the Société Anonyme de la Vieille Montagne, says: "In France, Westphalia and Rhenish Prussia, where a very fat coal is available, gas firing, with or without heat recuperation, is now generally adopted. In Belgium, on the other hand, where the coal is only semi-fat, direct firing is still generally maintained. Such Belgian works as have adopted gas firing are obliged to import German coal. The Société Anonyme de la Vieille Montagne has always sought to use the coal most easily obtained and consequently to avoid recourse to foreign coal retains direct firing for the more part of its furnaces."[1] In the United States up to the time of the establishment of the works in the natural gas fields direct firing was generally practised, the cost of coal in many of the zinc smelting districts being so low that there was

[1] The Mineral Industry, VIII, 655.

little incentive toward economy in its consumption by virtue of comparatively expensive recuperative furnaces. Such other advantages as gas firing might offer were undoubtedly lost sight of through the failure of the early attempts in Missouri and Kansas. At Lasalle and Peru in Illinois, however, gas firing was early adopted, and has continued in use. Neither in Europe or America has the Siemens regenerative system found any general application with the purely Belgian type of zinc distillation furnace; and so far as I am aware the Illinois Zinc Co., of Peru, Ill., is the only concern in the world which after a trial has retained it. This is not, however, because of an inapplicability of the Siemens system to the Belgian type, but rather because in building improved furnaces the Belgian smelters themselves are showing a tendency to abandon the characteristic Belgian type. Even the furnaces of the Illinois Zinc Co., although they are by no means new constructions, embody some of the important features of the Rhenish type.

In the earliest gas fired Belgian furnaces the width was increased so as to give room for flues in the middle wall. Two double furnaces were united in a block and at each end of the block were placed two generators. The products of combustion were led off through vertical flues in the middle walls to a horizontal flue communicating with a chimney placed centrally on the block. The furnaces were built longer than formerly but with fewer rows of retorts. Retorts of elliptical cross section also began to come into use. At Moresnet generators of the Boetius and Grobe-Lürmann type were employed; the former gave the better results.[1] Some furnaces were designed to distribute the gas to various parts of the combustion chamber,[2] and Loiseau proposed to make the temperature more uniform by introducing the air for secondary combustion at intervals, supplying cold air where the gas was richest in combustible and hotter air as the tenor in combustible decreased.

The gas fired furnaces at present in use in Belgium are the Angleur and Corphalie types, being thus designated according to the places where they are employed, and the Loiseau type, which is used at Ougrée and Bleyberg. In the United States, besides the Siemens furnaces used at Peru, Ill., the Hegeler furnace, which is employed at Lasalle, is the most important.

Loiseau Furnace.—This furnace, which was invented in 1878 by Oscar Loiseau, now director general of the establishments of G. Dumont et frères, was designed with the view of attaining a moderate and equable temperature through a series of combustion chambers, rather than an intense com-

[1] Wochenschrift des Vereins deutsch. Ingenieure, 1877, p. 14.
[2] Ibid, 1877, No. 44.

bustion and very high temperature. This was accomplished by introducing the gas in one division of the furnace and admitting the air required for its combustion at subsequent intervals, the air supply being preheated the more the less is the proportion of unburned gas remaining.[1] It will be observed that the same idea of the gradual introduction of the air supply obtained in the Hegeler furnace, which was patented about the same time; inasmuch as both patents have now expired the question of priority in the invention is of no consequence.

The Loiseau furnace as now used in Belgium has two working faces, the combustion chamber being divided by a middle, longitudinal wall, which does not rise so high as the roof arch. The gas supplied by an independent generator enters one division of the combustion chamber and thence passes over the middle wall into the other division, being fed gradually in the latter with air from ports in the middle wall, in which suitable ducts are arranged; in passing through them the air is heated at the expense of the combustion going on around the retorts on each side.

Loiseau furnaces are now in use at Ougrée and at Bleyberg in Belgium. At Ougrée, each furnace has six rows of 12 retorts per face, or 144 for the two faces. The retorts are elliptical, 0·18×0·20 m. inside, and 1·23 to 1·52 m. long. Each furnace is heated by a gas producer, outside of the furnace house, opposite one of the faces of the furnace. The second division of the combustion chamber communicates by means of a flue with a chimney on the other side of the house. The gas from the producer is burned incompletely in the first division of the furnace by a moderate admission of cold air. In the second division the partially burned gas mixes with preheated air issuing from the middle wall through ports at different levels. The retort residues are dropped from the face of the furnace opposite to the chimney through a slot in the floor into pockets, which communicate with a gallery below the floor. On the side opposite to the gas producer the canal from the latter prevents the arrangement of a long slot in the floor, and instead of the latter there are rectangular openings to the right and the left through which the residues are dropped. These openings are surmounted by sheet iron ventilating chimneys.

The Loiseau furnaces at Bleyberg are of similar design, having 2×6×12 =144 retorts, but two furnaces are combined as a massive (288 retorts), which is operated by a single chimney. Otherwise the arrangement is much the same as at Ougrée.

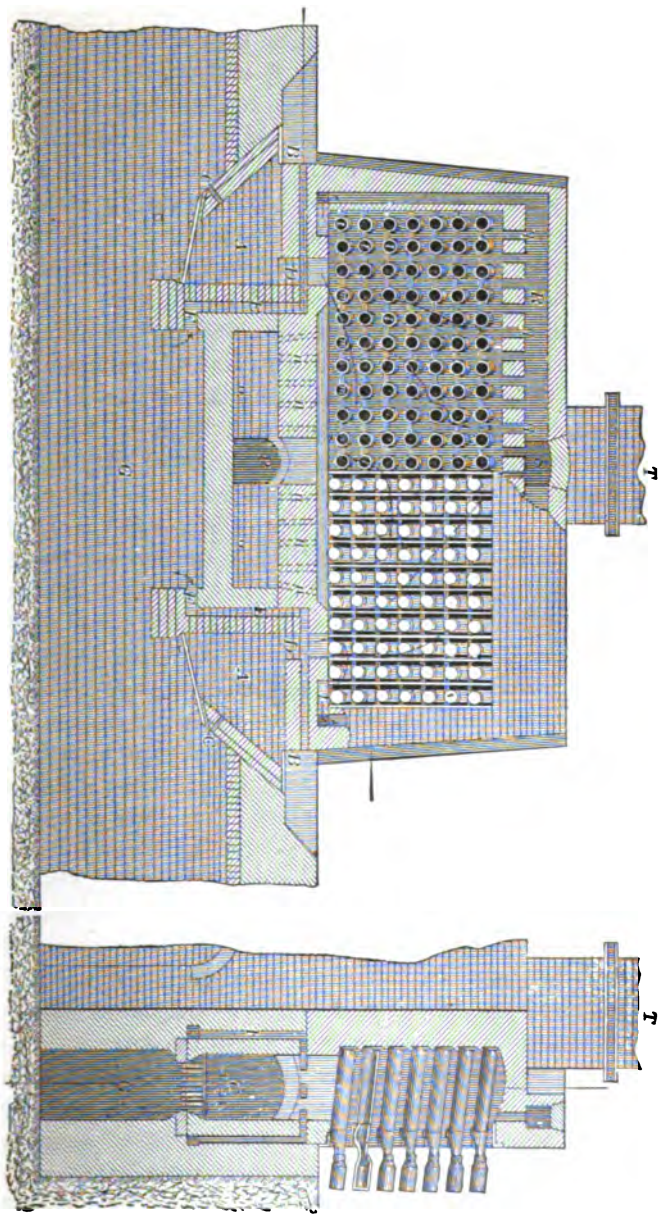
Corphalie Furnace.—The furnaces employed at the Austro-Belge works at

[1] Belgian patent of March 15, 1878 (application filed Feb. 22, 1878); Revue Universelle des Mines, IV, 303; Berg- u. Hüttenm. Ztg., 1879, p. 171.

Corphalie are peculiar inasmuch as they have single combustion chambers and only one working face. Two furnaces are united end to end to form a massive. Each furnace contains seven rows of cylindrical retorts, 0·20 m. in diameter inside and 1·26 m. in length (average). Each furnace has a generator set behind its rear wall and below the level of the floor. The gas reaches the combustion chamber through a short flue and burns with air preheated in ducts in the rear wall, whence it issues into the combustion chamber through ports at different levels between the rows of retorts. At Corphalie there are 42 of these furnaces, combined in 21 massives. The production of spelter in 1898 was 10,729 metric tons.

Angleur Furnace.—At the works of the Société Anonyme de la Vieille Montagne at Angleur the distillation furnaces have two faces, the combustion chamber being divided into two compartments by a longitudinal middle wall which does not, however, rise to the roof arch. Each side contains five rows of 20 retorts, in so far as most of the furnaces are concerned, but some of the more recent have only four rows of 20 each. The retorts are elliptical, the interior axes being 0·16×0·23 m. and the outside length 1·4 m. At the end of one side of the combustion chamber and under it, so that the charge opening is level with the floor of the furnace house, there are two ordinary gas producers, with grates. The products of combustion rise between the retorts of that side of the combustion chamber, pass over the middle wall and descend between the retorts of the other side. The air for completing in the second part of the furnace the combustion of the gas coming over from the first part is delivered through ports near the top of the middle wall, having been preheated by circulating through ducts in the latter. This method of heating is similar to that which is employed in the Hauzeur furnace (q. v.). At Angleur two furnaces are joined end to end to form a massive, which comprises therefore 400 or 320 retorts, according to the number of rows (four or five). There are 18 massives in the works, contained in three buildings, six in each, which are approximately 100×15·4 m., 100×14·75 m. and 100×17·6 m., the last being the most recent. The Angleur works produced 16,270 metric tons of spelter in 1898 and rank as the second largest in Belgium.

Boetius Furnace.—An early gas fired Belgian furnace with Boetius generators is shown in the accompanying engravings. This furnace has two combustion chambers and a generator at each end for each chamber, a total of four generators. Each chamber contains 147 retorts arranged in seven rows of 21, making a total of 294 for the furnace. The retorts are 1·25 m. long. The combustion chambers are 7 m. long, 1·2 m. wide, and 2·5 m. high. The generators are 1·2 m. deep and 0·93 m. wide. The flues con-



FIGS. 303 AND 304. BOETIUS BELGIAN FURNACE.

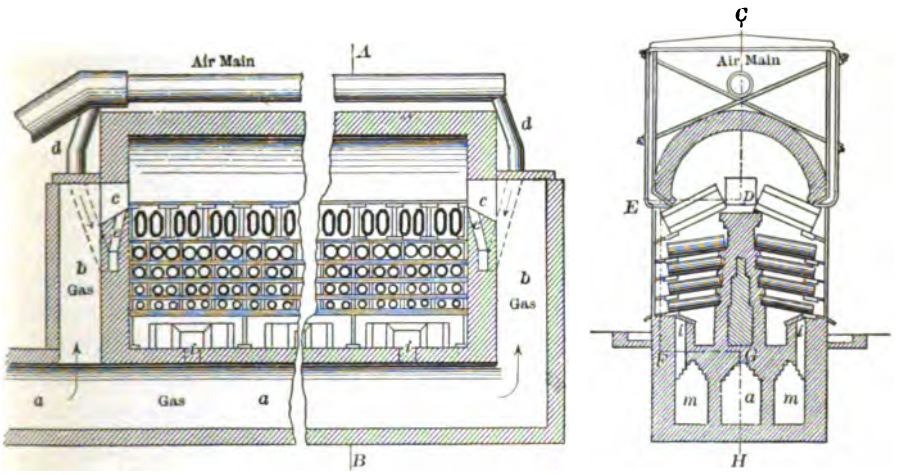
Fig. 303: Longitudinal section. Fig. 304: Transverse section.

necting the generators with the combustion chamber are 0·95×0·3 m. The generators are charged at *B* and kept constantly full of coal up to the charging door. The generators are of the regular Boetius type, with sloping grates. The air of secondary combustion enters from the tunnel under the furnace into the flues *E*, which surround the generator, and heated therein meets the gas at *D*. The combustion chamber connects by the openings *I* with a flue in each end wall, which flues rise vertically in the end wall and traverse longitudinally on the arch of the furnace, communicating with the chimney through *S*. There is also a longitudinal flue under the combustion chamber which communicates with the chimney through *P*, the flue of the chimney coming down through the middle wall of the furnace. The combustion chamber opens into the upper flue by the ports *qq*, and into the lower flue by the ports *nn*. The idea of this furnace was to produce a movement of the burning gas approximately as shown by the arrows Practical experience has shown, however, that it is unnecessary to go to such lengths to produce an even circulation of the flames around the retorts.

Hegeler Furnace.—The Matthiessen & Hegeler Zinc Co., of Lasalle, Ill., taking advantage of the inventive genius of Mr. Edward C. Hegeler, has been in a good many respects the most venturesome of all the concerns engaged in zinc smelting in America. It was he who first demonstrated that blende could be roasted successfully in mechanically raked muffle furnaces and he who showed that gas firing practically eliminated a restriction as to the height of a distillation furnace; only since it would have been inconvenient to build and operate a furnace so high as his system of firing permitted he built a very long furnace and turned the gas in at one end, which came to substantially the same thing as if he had built a furnace of equal height and turned the gas in at the bottom. This was accomplished by introducing the quantity of gas required to heat the whole furnace and burning it gradually by the introduction of air at intervals. The experience of many years at Lasalle has demonstrated that this can be done successfully. The only natural limit of length of furnace that I can see is that at which the volume of the gases of combustion would be so great that the dilution of the unconsumed gas by the inert gases mixed therewith would unduly retard chemical activity and the cross section of the furnace, i.e., the vertical dimension (because the horizontal is limited already by the length of the retort) would need to be increased to such an extent to permit the passage of the gases at a suitable velocity that the bounds of economy would be reached in that direction. As to whether these excessively long furnaces are good or bad practice because of other reasons will be discussed later on.

Hegeler's first large furnace, which was erected in 1872, had a long, high

arched combustion chamber with three longitudinal flues under the hearth. The middle wall did not rise so high as the chord of the arch and served only as a rest for the rear ends of the retorts. Of the latter there were five rows. The uppermost were vessels of octagonal cross section, 1·3 m. long, 0·50 m. high and 0·20 m. wide, inside measurements. The retorts of the lower rows were cylindrical, the diameters decreasing with successive rows. In the highest row there were 36 retorts; each of the other four rows had 42, so the total number per side was 204 and per furnace 408. These required a combustion chamber about 40 ft. long. The arrangement of the retorts and the peculiar method of taking up the thrust of the arch are shown in the accompanying engravings. The gas from a battery of step



FIGS. 305 AND 306. EARLY FORM OF HEGELER FURNACE.
Fig. 305 · Longitudinal section on line CDEFGH of Fig. 306. Fig. 306 : Transverse section on line AB of Fig. 305.

grate generators was led to the furnace through the canal *a*, which extended under the middle wall to the further end. At each end it rose through flues *bb* and entered the combustion chamber through *cc*, uniting there with air blown through pipes *dd*. The burning gas drew down between the retorts and escaped through ports *ii*, at intervals in the hearth, into the flues *mm* which led to the chimney. Those flues enclosing the gas flue *a* presumably gave up some heat to the incoming gas. At the time this furnace was in use it was the practice to distil two charges per 24 hours.

In the modern Hegeler furnace most of the peculiarities of the original type have been abandoned, including the high arch, retorts of different shapes, flues under the hearth, etc. There is now a simple combustion cham-

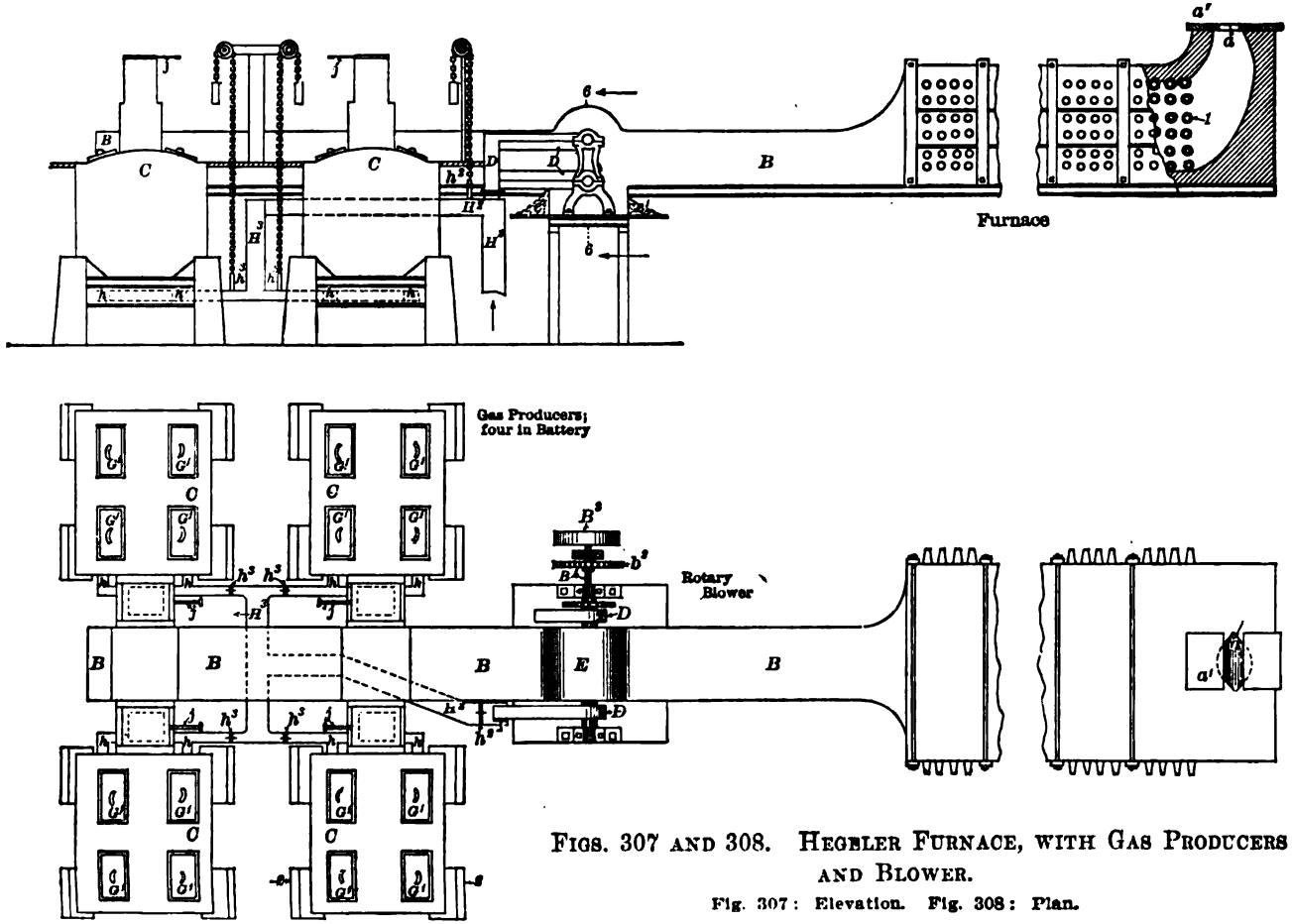
ber, with a low arch springing from one of the long side walls to the other, and a middle wall which rises to the arch and divides the combustion chamber into two sections. Under the hearth there is a tunnel with which the ash chutes along the working faces of the furnace communicate. The middle wall stands on the arch of the tunnel. At one end of the furnace the gas from a battery of generators is turned in, a short room being left clear in advance of the retorts to enable the gas to become thoroughly mixed with the first supply of air. The generators have been blown with an undergrate blast alone, but in 1900 one of the furnaces was equipped with an exhauster interposed between it and the producers. The latter are separate from the furnaces, but only a few feet away, so the gas arrives with much of the heat of primary combustion. Extending longitudinally over the furnace is an air main from which branches are brought down on each side, there being a branch between every four retorts; consequently if it be a six-row furnace each side is thus divided into sections of 24 retorts. Between each section the pillars of the front wall are made wide enough to permit air ports to penetrate them and the air pipes are kept far enough away from the condensers not to interfere with the tapping of the latter. The air is blown into the furnace under pressure of 1·5 in. of water. The furnaces now in use have $2\times56\times4=448$ retorts, $2\times72\times4=576$, $2\times72\times6=864$, and $2\times84\times6=1008$, the furnace with 1008 retorts being the one most recently constructed. The products of combustion escaping from the far end of the furnaces are led through a recuperator, which is essentially a brick chamber containing a series of iron pipes through which a current of air is blown. The air main over the furnaces gets its supply from these stoves. The preheated air does not reach the furnaces at a high temperature, however, being conveyed through an iron main from which a good deal of heat is naturally radiated. Anyway the conduction of the air through branch pipes outside of the furnace would preclude it from being heated above a moderate temperature. The gases escaping from the recuperative stoves are still so hot that they are available for making steam and batteries of boilers are connected with some of the furnaces. In that way sufficient steam is generated to supply not only the entire smelting works but also the large rolling mill that is connected with them. The combined system of stoves and boilers makes a well developed system of heat recuperation. Several furnaces together with their gas producers, stoves and boilers are contained in one huge house. On each side of each furnace there are tramway tracks for ore charging cars, metal drawing cars, etc., for the convenient working of the furnaces. Such labor saving devices have been developed more extensively at Lasalle than anywhere else.

In the latest installations of the long distillation furnaces at Lasalle, Ill., a rotary exhauster designed on the principle of the well known Roots blower has been interposed between the battery of gas producers and the distillation furnace proper. In order to operate the furnaces to the best advantage the gas must be at a uniform pressure above atmospheric pressure at the entrance of the combustion chamber and must have a uniform temperature, so far as practicable, without losing any of its sensible heat. In the older installations the feeding and stoking openings of the producer had to be kept closed in order to obtain the requisite pressure at the entrance into the combustion chamber and when feeding and stoking were necessary the blast under the producer had to be interrupted or reduced, so that the gas would not drive through the openings toward the operator; in removing the ashes the blast had also to be interrupted. These interruptions led to reduction of the pressure and quality of the gas and when the feeding and stoking holes were open more or less air would naturally enter through them, either burning the gas and producing a high temperature in the gas flue, or if the temperature of the gas were below the ignition point leading to danger of explosions in the gas flue. In the new installations it is possible to cut any one of the producers out of the battery during feeding or stoking by an arrangement which is shown diagramatically in the accompanying engravings and more clearly in Chapter VIII, where the Hegeler producer is also described and illustrated in detail.

The improved system permits an approximate equilibrium between the interior pressure of the producer and the exterior atmospheric pressure at the feeding and stoking openings to be produced whenever desired, so that neither blowing out of gas nor drawing in of air sufficient to cause burning in the gas flue will occur when the openings are uncovered, and at the same time without interfering with the pressure at the entrance of the combustion chamber. This enables an approximately uniform movement of the gas toward the combustion chamber at a nearly regular temperature to be maintained, the necessary poking and feeding to prevent dwindling of the gas being made possible without interrupting the operation of any producer.[1]

In the accompanying engravings *B* is the flue connecting with the producers, and *CCCC* are the gas producers arranged four in a battery. The combustion chamber is shown as if built directly on the ground, but beneath it there is a longitudinal arched tunnel, with which connect the ash chutes into which the retort residues are discharged. The gas flue communicates with the combustion chamber at one end and the gas passes through the space between the retorts in the combustion chamber, where it is burned

[1] U. S. patent No. 630,885, Aug. 15, 1899.

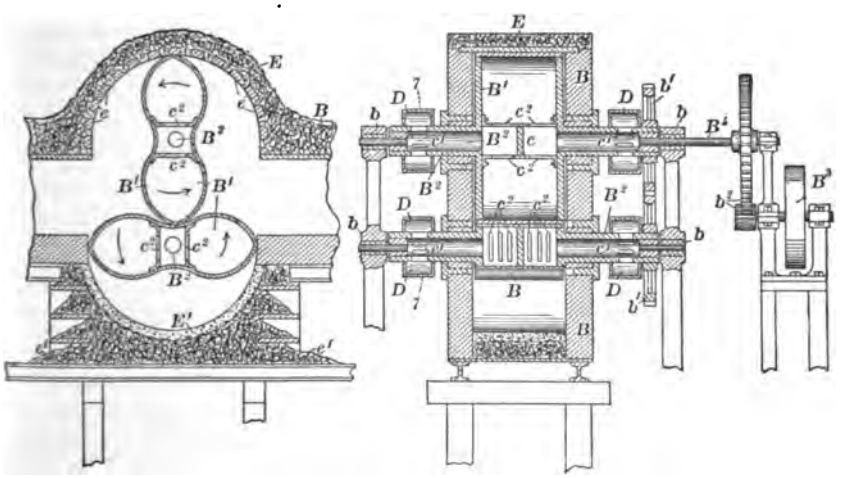


FIGS. 307 AND 308. HEGELER FURNACE, WITH GAS PRODUCERS AND BLOWER.

Fig. 307: Elevation. Fig. 308: Plan.

in the ordinary way, the burned gases passing out at an opening *a* in the top of the combustion chamber, over which a sheet iron chimney is suspended in a manner to allow air to enter it at the bottom for producing a cooling effect in the chimney. The opening *a* may be more or less closed for regulating the pressure in the combustion chamber by fire clay bricks or slabs *a'* adapted to slide over the same.

Rotary Blower for Hot Gas.—A rotary blower, or exhauster, is placed in the gas flue between the gas producer and combustion chamber for drawing the gas from the producer and forcing it into the combustion chamber. This exhauster is run at uniform speed and produces a uniform movement of gas in the flue, automatically retarding or promoting movement in the producer,



FIGS. 309 AND 310. ROTARY BLOWER FOR HOT GAS.

Fig. 309: Transverse section through impellers. Fig. 310: Longitudinal section through impellers.

accordingly as there is a tendency to produce more or less than the required amount of gas. The exhauster in the gas flue therefore coöperates with the blast beneath the fire to lessen or promote the production of gas in accordance with the quantity the exhauster takes away, and when used with the means for increasing or diminishing the draught openings beneath the fire, so as to admit more or less air, according as there comes less or more than the amount taken away by the exhauster, the required equilibrium in the upper part of the producer may readily be secured and maintained while the feeding and stoking openings are uncovered.

The exhauster comprises wings B^1, supported on the shafts B^2, which are

extended transversely through an enlarged part of the flue adapted to the wings and supported in suitable bearings b. The wings are operated in unison by spur-wheels b' on the shafts B^2. One of the shafts is connected with suitable gearings b^2, adapted to be operated by a pulley B^3, which is driven by a belt from the power shaft. In order to secure the exhauster against accidental high heats and for keeping the shafts, spur-wheels, journals, and bearings cool the shafts are made hollow in that part which passes through the gas flue and for some distance upon each side. There is a partition c midway in the hollow shaft. At the ends outside the flue there are slots or openings c' communicating with the hollow of the shafts. The wings are also made hollow, and the openings c^2 in the shafts upon each side of the partition c are made to communicate with the hollow of the wings. Hollow boxings D are placed upon the shafts so as to cover the openings c' and a blast pipe D' is connected with the boxings, whereby a current of cold

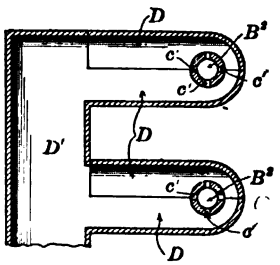


FIG. 311. ROTARY BLOWER FOR HOT GAS.
Section through air-boxes and shafts, showing means for cooling the latter.

air may be forced in at one side by an ordinary blower, passing through the hollow of the shafts and out upon the opposite side.

The wings of the exhauster rub, or move relatively to one another, at the point of approximate contact, and thereby cleanse each other of all accumulations of soot upon their adjacent faces, except to the extent of such small increment as will make them fit more closely together; such soot as remains on them being a non-conductor tends to protect the exhauster by preventing absorption of heat from the gas.

The soot accumulations upon the interior of that part of the flue where the exhauster is located are subject to being forced outward against the wall by the action of the wings, and thus tend to produce objectionable pressure upon, or contact between, the wings and flue. To obviate any difficulty on that account, a yielding section is arranged in the bottom and top of the flue where the exhauster is located, which will give way before any dangerous

pressure is created in that manner, and also in case of accidental explosions. Such a safety-valve is provided by forming part of the arch *E*, over the wings, of cross-bars *e* laid sufficiently close to support a layer composed of a mixture of clay and coal dust, and of sufficient weight and strength to resist the gas pressure and yet yield to any outward pressure resulting from soot accumulations being forced outward by the action of the wings of the exhauster. The bottom part *E'* is made of a layer of similar composition supported on a bed of dust *e'*, piled upon the floor under the gas flue. These yielding parts of the flue not only adapt it to utilize the soot accumulations for maintaining a close fit of the wings in the flue, but also afford means of easy access to the interior of the flue for cleaning or repair. When the exhauster in the gas flue is out of repair, the gas is carried around it by means of a by-pass, while the repairing is being done. The battery of producers in such case may be worked by means of the valves controlling the communication of the several members with the gas flue and the escape chimney, whereby any single producer may be cut out during the feeding, stoking and clinkering thereof, the other producers of the battery furnishing the supply of gas in the meanwhile, the blast beneath the fire being of sufficient strength to be turned on with increased force at the same time.

Advantages and Disadvantages of Long Furnaces.—I have no data as to the technical and commercial results of the Hegeler furnaces. The fact that they are successful at Lasalle, as to which there is no doubt, does not imply that they are the best, which could be determined only by a practical trial against some other type of furnace under identical conditions. Their design involves, however, certain theoretical principles which are capable of discussion. (1) In the first place as to consumption of fuel: There cannot possibly be any economy as compared with a well designed furnace of another type, because the products of combustion discharged finally from one long furnace must be of the same volume and temperature as from an equivalent number of shorter furnaces, i.e., the heat lost in the chimney gases would be the same in either case; the number of end walls being smaller there is less radiating surface exposed, but the economy in that direction would be insignificant; on the other hand it is likely that the combustible gas which reaches the last sections of the furnace unburned is so much diluted by the great volume of carbonic acid and nitrogen from the previous combustion that its chemical activity is reduced, which would imply imperfect combustion; at all events combustion is purposely retarded in other processes in precisely that manner. From theoretical considerations it would appear, therefore, that extremely long distillation furnaces would not tend toward the maximum economy in fuel, but the contrary.

There is an advantage, however, in the arrangement of the gas producers for a large furnace that must not be overlooked. The furnace will require so much gas that no single generator can supply it. There must be, therefore, of necessity a battery of generators to each furnace, and such battery can be placed close to the furnace, thus preserving much of the heat of primary combustion of the coal and at the same time affording the even supply of gas that can be maintained only when several generators are grouped in a battery. It is also more economical to deliver coal to them and manage the producers. These advantages are not, however, necessarily restricted to excessively long furnaces. (2) Labor: Long furnaces are charged from end to end, the most easily reducible material being put in the retorts next to the chimney end. The charging is done according to tiers of retorts instead of by rows. This method, which obtains also with the Iola furnaces, permits of a better systematization of labor, aided by the application of labor saving devices which is possible with long furnaces. There is no doubt that long furnaces lead to an economy in labor. (3) Extraction of metal. Assuming that the labor be so systematized that the charged retorts are exposed to the necessary heat for the requisite length of time, the percentage of metal extracted from the ore will depend largely upon the ability to maintain the proper temperature toward the chimney end. There is no doubt that the long furnaces run cooler there than toward the gas inlet end. However, the retorts of the chimney end are naturally charged with the more easily reducible material, such as blue powder, skimmings, etc. (4) First cost: There is undoubtedly a saving in first cost per retort the longer the furnace is, not only in material and labor, but also in floor space and building to cover it. On the other hand there is the disadvantage that in putting a furnace out of operation for repairs the capacity of the works may be immoderately reduced. Thus, if a plant comprised 2000 retorts divided between two furnaces, the repairing of either one would reduce temporarily the output of the works by 50%, while if the same number of retorts were divided among five furnaces the stopping of any one would cause a reduction of only 20%.

Producer Gas Fired, Recuperative.

Gas-fired Belgian furnaces are to be found with heat recuperation by three systems, viz., the reversing, with checker-works for both air and gas, as in the Siemens-Belgian; reversing, with checker-works for air alone, as in the Ferraris, and continuous, or counter-current, as in the Convers and De Saulles. In most of the furnaces of this type the regenerative chambers are arranged

directly beneath the combustion chambers. In the case of open-hearth steel furnaces the construction wherein the working hearth is founded on the arches of the alternately heating and cooling regenerative chambers has been severely criticized, but in this respect the zinc distillation furnace is different, the hearth of the combustion chamber having no function to perform. However, the Dor furnace, which has been deemed worthy of extended description herein because of the novelty of its idea and the eminence of its

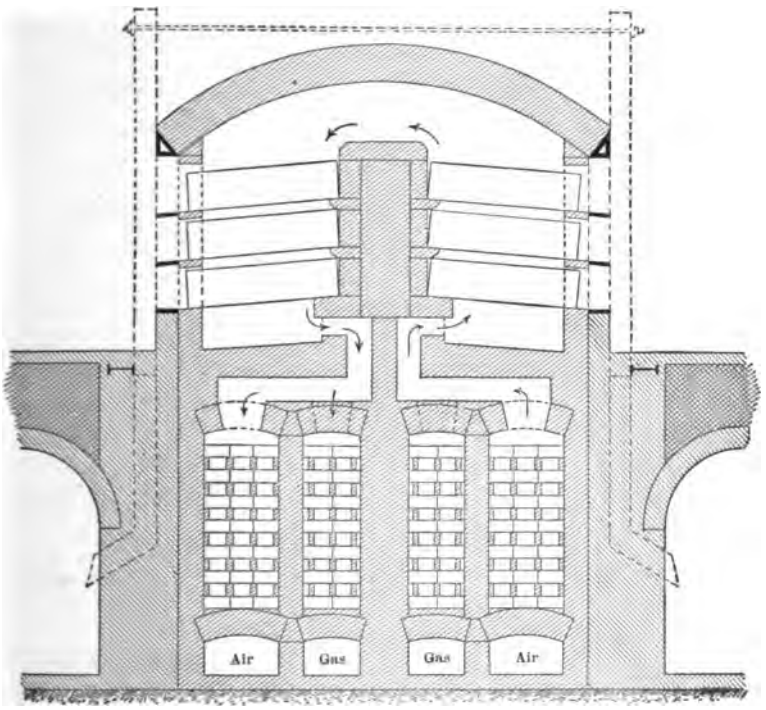


FIG. 312. SIEMENS-BELGIAN FURNACE.
Transverse vertical section.

inventor, although it did not come into regular use, shows vertical regenerative chambers at each end. The gas fired, recuperative Belgian furnaces, which with a single exception, are comparatively recent designs are characterized by many features taken from the Rhenish type, so that their proper classification is often doubtful.

SIEMENS-BELGIAN FURNACE.—The Siemens system of regenerative gas firing lends itself easily to the Belgian distillation furnace, the regenerative chambers being installed in rooms under the hearths of the combustion

chambers. The typical arrangement is shown in Fig. 312, which does not require an extended description. The reversing valves, not shown, are at the end of the massive. The gas and air enter the respective regenerative chambers of one side of the furnace, and from them are delivered through alternate ports into the overlying division of the combustion chamber, beneath the lowest row of retorts. Passing over the longitudinal wall into the other division, the flames envelop the retorts therein and escape through the ports of that side into the corresponding pair of regenerative chambers and thence to the chimney. At the end of 30 minutes the directions of currents are reversed and so on.

For various reasons the Siemens furnace did not meet with much favor in Belgium until recently.[1] They were installed at Overpelt in 1893, at

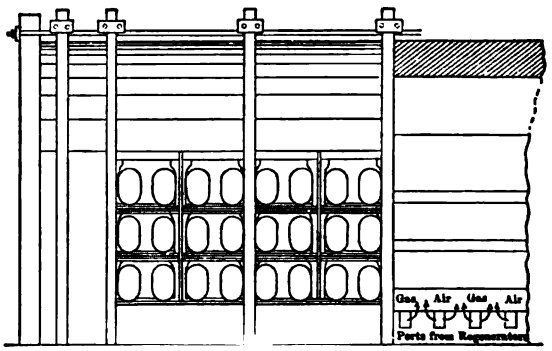


FIG. 313. SIEMENS-BELGIAN FURNACE.

Front elevation and longitudinal section through ports communicating with the regenerative chambers.

Prayon in 1894, and at Engis. In 1898 and 1899 additional massives were erected at both Prayon and Engis to replace older types of furnaces. All of those constructions have partaken, however, of the Rhenish form of furnace, rather than the purely Belgian, and they have previously been described under that caption.

In the United States there have been three installations of Siemens-Belgian furnaces, namely by the Granby Mining and Smelting Co. at Pittsburg, Kan. (built in 1882); the Rich Hill Mining and Smelting Co. at

[1] Some of the objections that have been made against the Siemens furnace are its high first cost and subsequent large expense for repairs and renewals, the higher class of labor required for its successful management, its production of a higher temperature than is required for the distillation of the charges in small Belgian retorts, and difficulties in its operation, among others the choking of the checkerworks by accumulations of zinc oxide volatilised from leaky retorts.

Rich Hill, Mo. (built about the same time as the Granby works), and the Illinois Zinc Co. at Peru, Ill. The last is the only one of the three which continues to use them.[1] The Siemens-Belgian furnaces which have been used in the United States have had certain points of resemblance to the Rhenish type of furnace, especially in having only a few rows of retorts and the retorts of comparatively large elliptical section. The deep niches and other characteristic details of that type have not, however, obtained. The furnace at Rich Hill had three rows of retorts per side, 48 in a row, making a total of 144 per side and 288 for the furnace. The furnace at the Granby works had $2\times4\times50=400$. At Peru, Ill., there are now furnaces with $2\times4\times56=448$, $2\times4\times60=480$ and $2\times4\times68=544$; but the original furnaces at Peru were of the same dimensions as those at Rich Hill, Mo. The retorts at Peru are 51 in. long and $11\times7\cdot5$ in. diameter inside. Those at Rich Hill and the Granby works had about the same dimensions. The furnaces at each of these places were essentially like that illustrated in Fig. 312. The $2\times3\times48=288$ furnaces at Rich Hill and Peru were about 60 ft. long. The regenerative chambers were 28 in. wide, the aggregate checkerwork comprising 30,000 brick. These furnaces would receive a charge of 16,000 to 17,000 lb. of roasted ore. The installations were made with the old form of Siemens generators, with step grates and siphon draught pipe. At Peru, however, the gas is now produced by means of Taylor generators.

Both at Rich Hill and at the Granby works the Siemens furnaces were failures. So far as I can learn the trouble was with the gas supply rather than with the furnaces, the foreman of the Rich Hill works informing me (many years after the Siemens furnace there was abandoned) that it was impossible to heat the furnace properly because of lack of gas. It is generally conceded, I believe, that the Siemens gas producer was never a good one, the widespread use it found being due to the excellence of the Siemens regenerative system. The coal at both Rich Hill, Mo., and Pittsburg, Kan., is a badly clinkering one which would give trouble even in a good generator and it is not surprising that poor results were obtained with the Siemens. At Peru, Ill., the Siemens furnaces appear to have been a success.

Charles F. Neureuther, of Peru, Ill., patented a modification of the Siemens regenerative Belgian distillation furnace, in which the air and gas are partially discharged half way up in the combustion chamber by means of vertical ducts and corresponding ports in the middle longitudinal wall,

[1] The Granby works have been abandoned and torn down. The Rich Hill works have been lately operated under lease by the Cherokee-Lanyon Spelter Co., which employs direct fired Belgian furnaces, the Siemens furnace being dismantled.

which arrangement is illustrated in the accompanying engraving. In the Siemens regenerative Belgian furnaces built heretofore in the United States the gas and air have been discharged under the lowest row of retorts; with that method of heating, according to Neureuther, the retorts of the upper

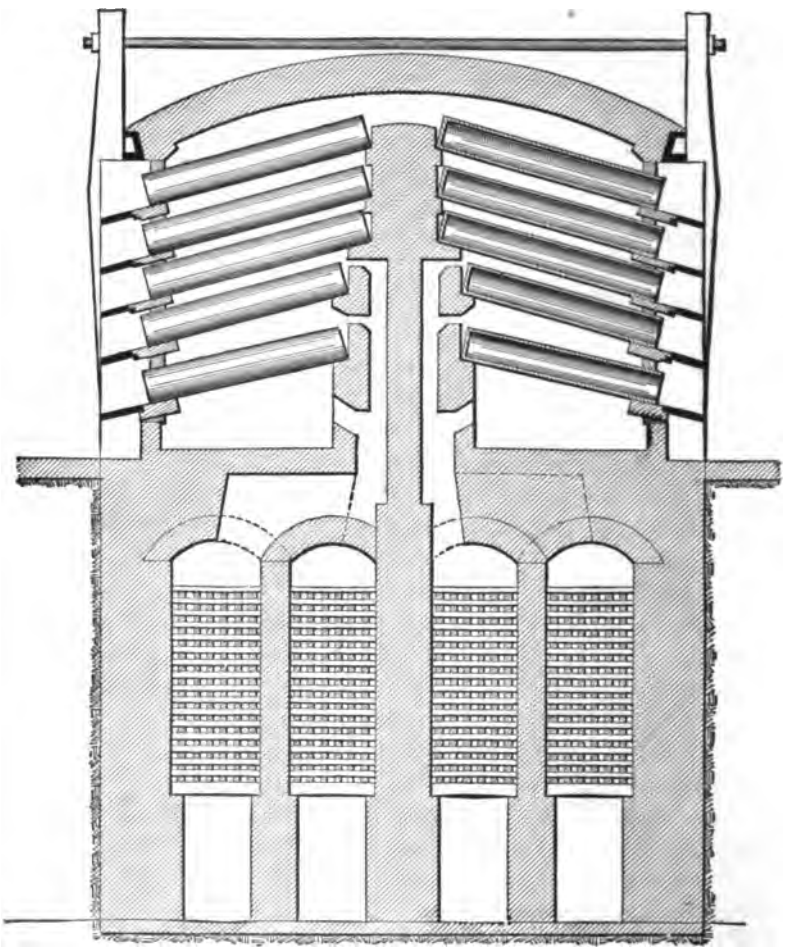


FIG. 314. NEUREUTHER SIEMENS FURNACE.

rows have not been so highly heated as those of the lower, with the result that the ore charged into the upper retorts has required a longer time for distillation and the residues have been richer in zinc than they ought to be. He claims that by introducing the gas and air at different points in the

height of the combustion chamber a more uniform temperature is obtained; also that the capacity of the furnace is increased and that it is practicable to install at least one more row of retorts than in the older furnaces.[1] The building of ducts in the middle of the longitudinal wall necessitates that the latter be made thicker at the bottom, on which account the retorts of the lowest two rows are made necessarily shorter than those of the upper. This furnace is in use at the works of the Illinois Zinc Co.

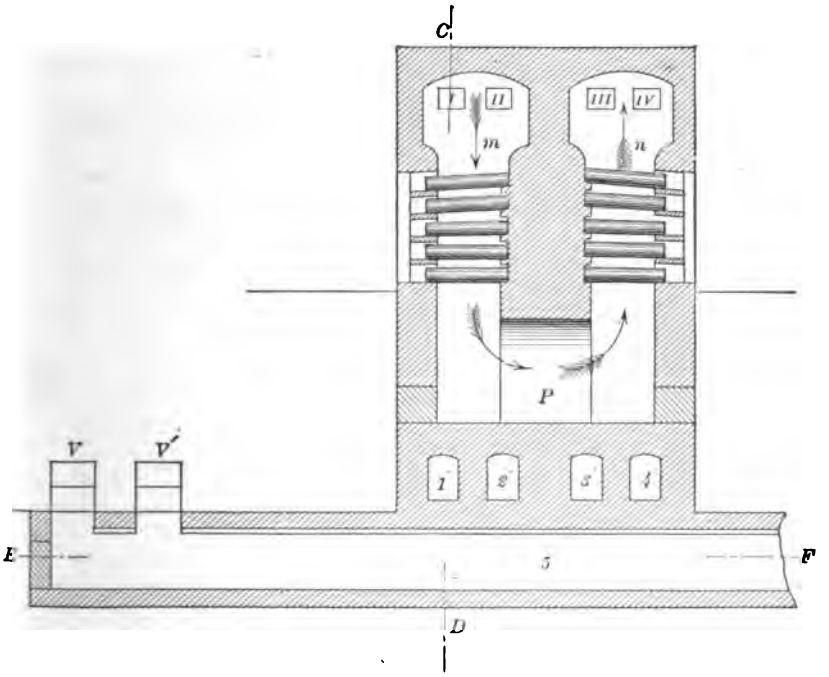


FIG. 315. DOR FURNACE.
Section on line AB of Fig. 316.

DOR FURNACE.—Emile Dor, now deceased, who was for many years director of the Laminne works at Antheit, Belgium, patented a regenerative furnace in which the heat recuperating chambers were placed at the ends of the furnace instead of beneath the combustion chamber as ordinarily.[2] As shown in the patent specifications the two combustion chambers of the furnace were characterized by large free spaces above and below the ranks of the retorts. At the bottom a large, arched opening through the middle wall

[1] United States patent, No. 666,390, Jan. 22, 1901.
[2] British patent, No. 22,694, of 1891.

afforded free communication between the two chambers. At each end of the furnace there were four vertical chambers, subdivided into flues by suitable brick work, each one communicating with the combustion chambers by ports through the end walls of the latter. There were, therefore, at each end of the furnace an air and a gas regenerator for each side. The air and gas were delivered to the furnace through canals entering at right angles to the long axis of the furnace and discharging into ducts parallel with the long axis, all of these flues being beneath the combustion chambers, and thence passing through the regenerators. Of course the products of com-

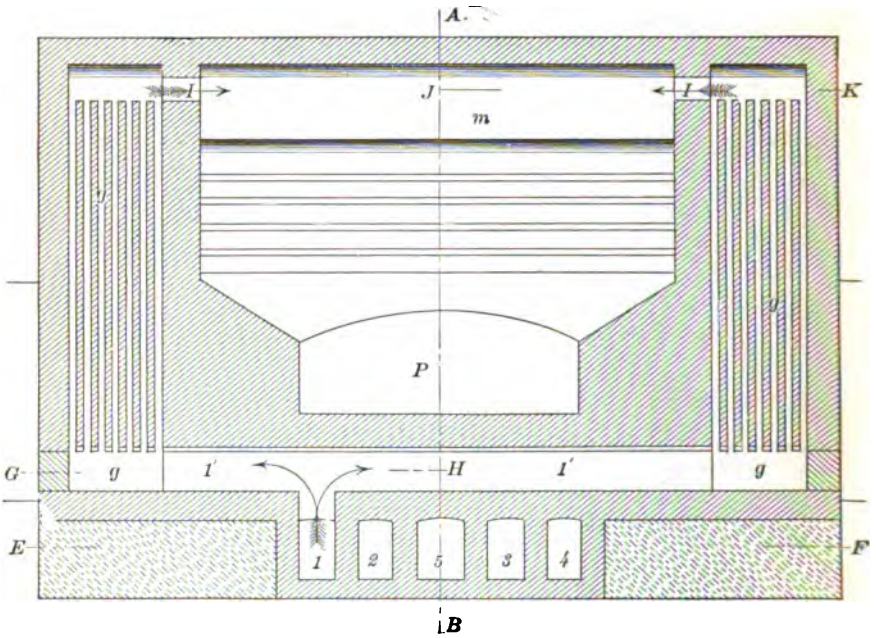


FIG. 316. DOR FURNACE.
Section on line CD of Fig. 315.

bustion escaped in the opposite direction; the reversal was effected in the ordinary manner.

In operation, the gas and air arriving through their separate canals were divided, one half of each going to one end of the furnace and the other half to the other end. Rising through the appropriate regenerative chambers, in which it will be observed the movement of the gases was vertical, the then hot air and gas were discharged into the combustion chamber of say the left-hand side. The burning gas passed downward between the retorts

of that side, through the large opening in the middle wall into the combustion chamber of the right-hand side, upward between the retorts of that side, and thence through the corresponding regenerative chambers and flues to the chimney. After a suitable interval the currents were reversed, the gas and air then passing first through the right-hand combustion chamber and thence into the left-hand side.

I am ignorant as to whether this furnace, which it will be observed is of quite novel design, was ever put into practical use. This furnace is not to be confused with the modification of the direct fired Liège type which is also known as the Dor furnace and is now in use at Antheit and to some extent

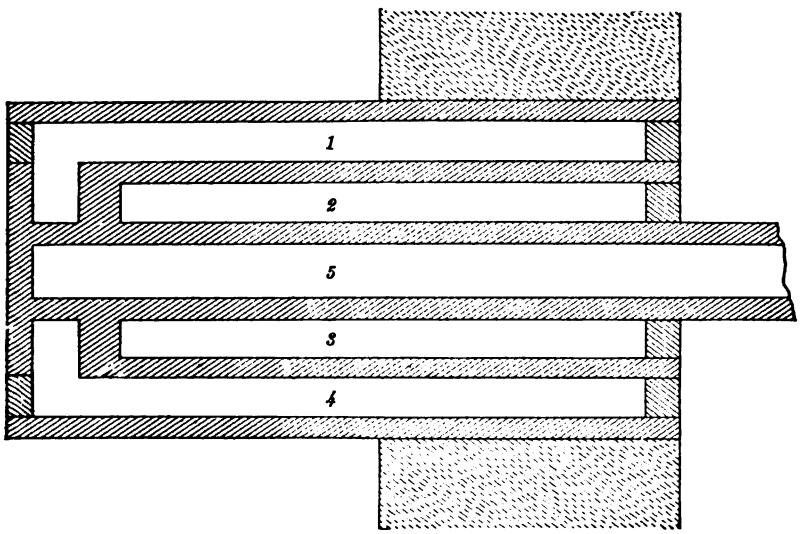


FIG. 317. DOR FURNACE.
Horizontal section through main gas and air canals.

also at Ougrée, Bleyberg and Engis. These furnaces are direct fired, with step grates of which there is one for each single furnace. The single furnaces at Antheit have 50 retorts, arranged in five rows. Four furnaces are grouped in a massive, which has consequently 200 retorts. The retorts are elliptical, 0·178×0·209 m. in diameter and 1·35 m. in length.

A good deal of the recent progress in the metallurgy of zinc in Belgium is due to M. Dor. He invented the hydraulic presses for making the retorts, which are now so generally used, and was one of the first, if not the first, to introduce the elliptical form of retorts and to reduce the height of the furnace to five rows.

FERRARIS FURNACE.—This furnace, invented by Erminio Ferraris,[1] has lately been introduced at the works of the Società di Monteponi, in Sardinia, where it has given very successful results. It is gas fired and heat recuperative, but has regenerative chambers for air alone, the gas being delivered already hot from the generator (which should be therefore set as near the furnace as is convenient) and being further preheated only by cir-

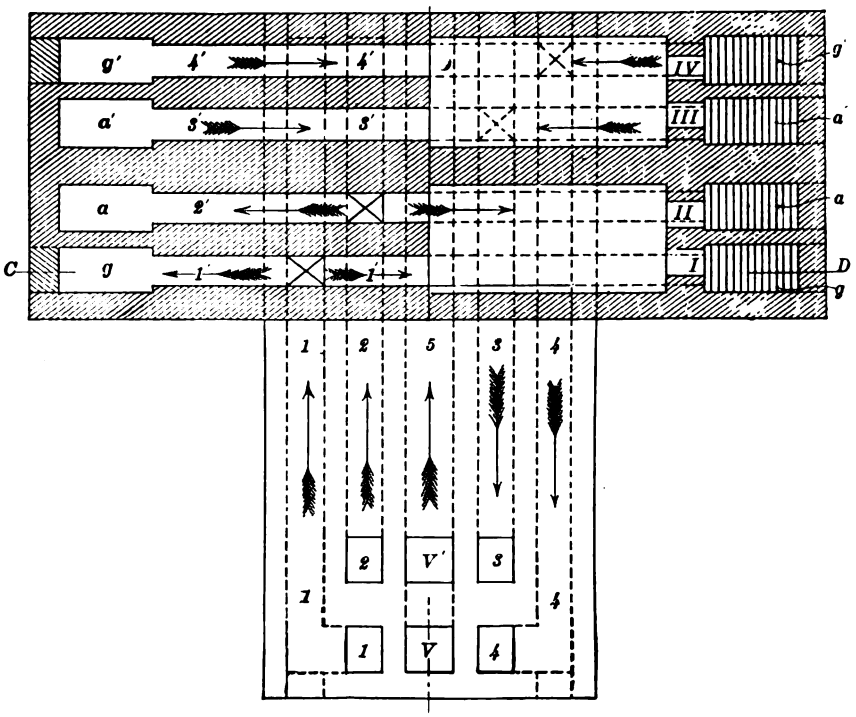


FIG. 318. DOR FURNACE.
Horizontal sections showing courses of air and gases.

culation through flues in the middle longitudinal wall, which divides the furnace into two separate combustion chambers. The movement of the burning gas through the combustion chambers is longitudinal, but the direction of the current is periodically reversed, wherefore there is no cold end. Beneath each combustion chamber there are two heat recuperative chambers, filled with hollow brick, standard fire brick, or tiling, in such a way as to form numerous horizontal, parallel flues, which are divided into two groups by a horizontal partition, compelling the products of combustion entering

[1] United States patent, No. 714,685, Dec. 2, 1902.

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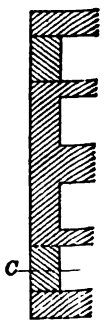

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at one end of the recuperative chamber from the corresponding end of the combustion chamber to traverse horizontally through the recuperative chamber, first through the upper division toward the middle of the furnace and thence backward through the lower division to the end, where communication is made with a longitudinal flue leading to a main flue which takes off from the middle of the furnace to the chimney.

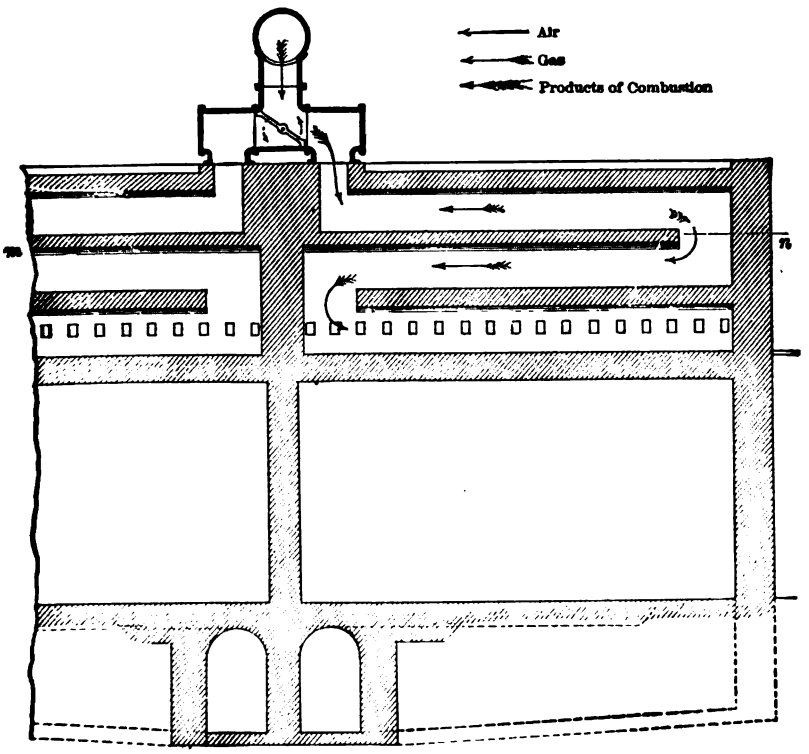


FIG. 321. FERRARIS FURNACE.

Vertical section through gas flues, on line *hi* of Fig. 323.
Scale, 1 : 100.

The gas from the producer, situated at one side of the furnace house, is delivered through a properly lined pipe to the center of the furnace overhead, whence it can be directed by changing the position of a valve alternately to one end of the furnace or the other, passing through the flues in the middle wall. It enters the combustion chamber through ports below the lowest row of retorts, which ports are provided at intervals up to a line

midway and transversely through the combustion chambers, so that the gas, instead of merely entering at the end, is distributed through the latter, thus insuring complete combustion of the gas by gradual admixture with air and a proper heat in the middle of the combustion chamber, which might otherwise average a little cooler than the ends. In the flue leading to the chimney there is a reversing valve to change the current of the products of com-

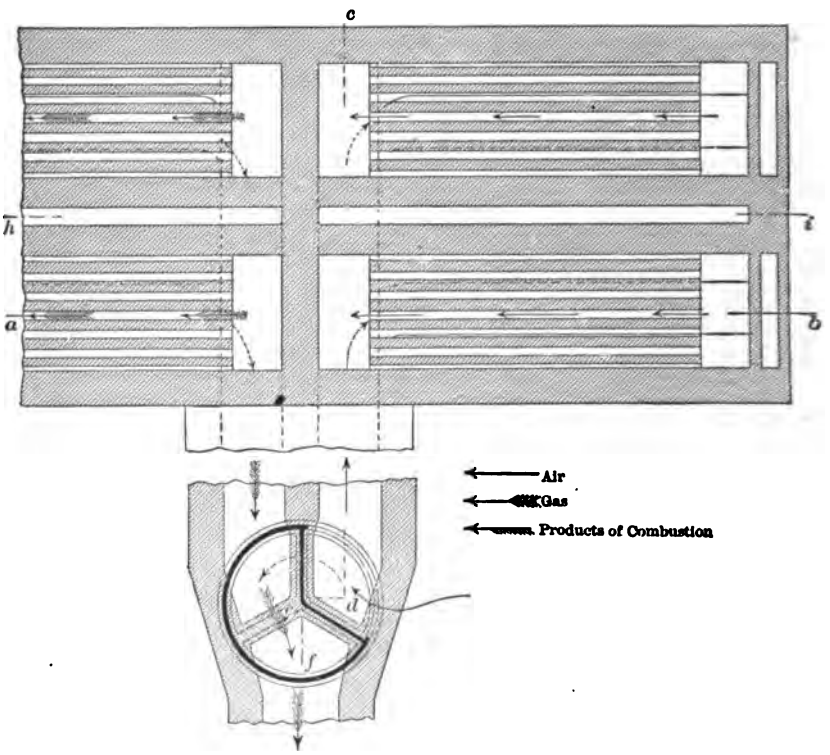


FIG. 322. FERRARIS FURNACE.
Horizontal section on line *opqr* of Fig. 319.

bustion, which valve is connected with the gas valve over the furnace, so that the entire system will be operated together.

In operation the gas entering at one end of the furnace meets the hot air issuing from the recuperative chambers corresponding to that end. The burning gas passes horizontally toward the other end of the combustion chambers, enveloping the retorts on the way, and thence escapes into the recuperative chambers corresponding to that end, traveling horizontally

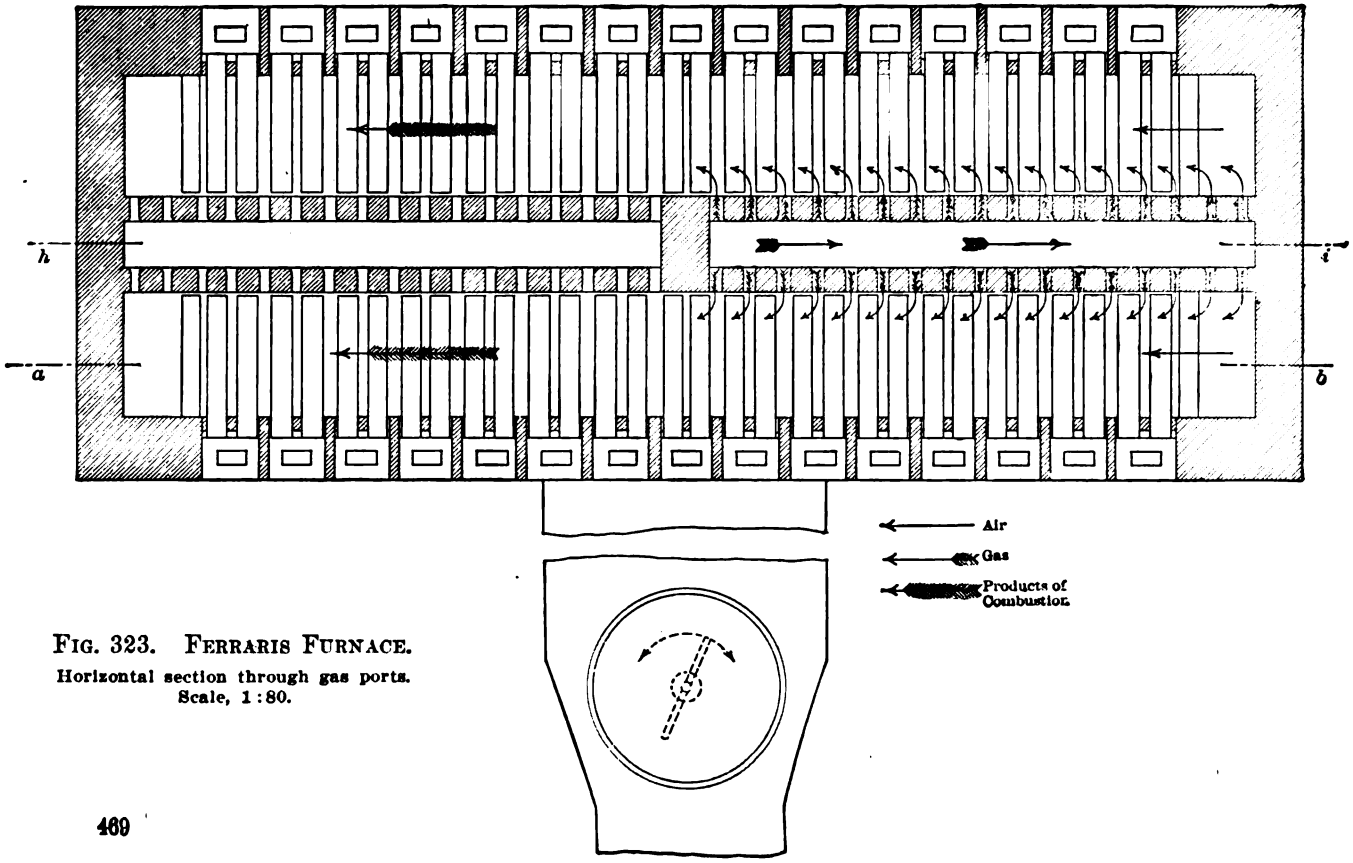


FIG. 323. FERRARIS FURNACE.
Horizontal section through gas ports.
Scale, 1:80.

through their flues and making a return pass before reaching the large flue leading to the chimney. In the meanwhile the air for the combustion of the gas is entering through the flues of the opposite pair of recuperators. At the end of the proper interval the direction of the currents of air and gas is reversed.

This furnace, which is designed on thoroughly scientific principles, has the important advantages of maintaining an equable temperature throughout a great length, and at the same time permitting the installation of a heat recuperative system of high efficiency. The appropriation of the entire available space under the combustion chambers to regenerative chambers for air alone simplifies the construction as compared with Siemens furnaces, in which heat is recuperated through the media of both gas and air. With regenerators for air alone there is no trouble from gas leakages and the hot brick of the flues is subjected only to the strongly cooling action of cold air. The gas, which is delivered to the furnace from above, arrives with much of the heat of the primary combustion in the producer, and is further heated during its circulation through the flues in the middle longitudinal wall; the heat that is acquired in the latter manner is no actual gain, however, inasmuch as it is abstracted by conduction and radiation from the combustion chambers to which it is immediately returned as caloric of the gas. The furnaces at Monteponi have 180 large retorts per massive and smelt from 6000 to 7000 kg. (13,228 to 15,432 lb.) of calcined ore per 24 hours. The fuel employed is lignite of low calorific power.

Convers & De Saulles Furnace.—This furnace, patented by George G. Convers and Arthur B. De Saulles,[1] is a highly developed example of a distillation furnace designed for the counter-current system of heat recuperation, in which not only is the air preheated, but also the gas to some extent. The construction of the furnace is shown in Figs. 324 to 328. It has a single combustion chamber, in which the inner ends of the retorts are supported in much the same way as in the Rhenish type of furnace, while the circulation of the flames is similar, but in the reverse direction, the products of combustion escaping through a shaft on the middle line of the furnace, between the ranks of retorts on each side. The admission of gas and air takes place through ports, which are similar in arrangement to those of the Siemens-Belgian furnace.

The gas from the producers enters through the canal *H*, passing thence through the uptake *I* into the canals *JJ*, from which it is discharged into the combustion chamber through the ports *d*, shown in Fig. 327 and by the dotted lines in Fig. 324. The air enters through the canal *E*, passing into

[1] United States patent, No. 712,502, Nov. 4, 1902.

the flues *DD*, and thence rising through the spaces between the flues in the recuperative chambers *FF* into the canals *GG*, whence it is distributed into

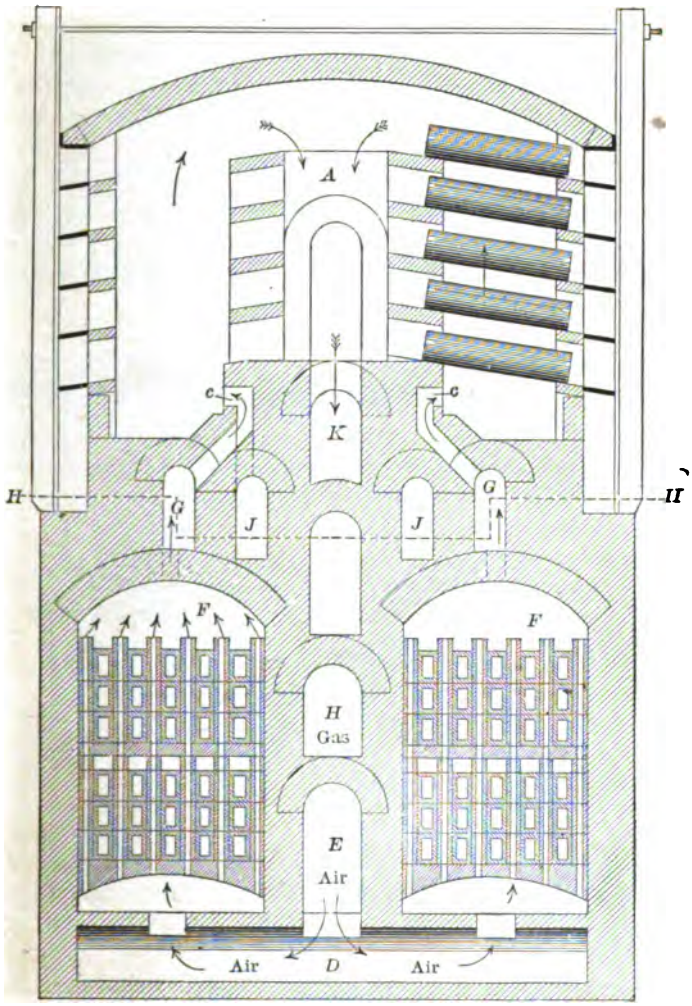


FIG. 324. CONVERS & DE SAULLES FURNACE.
Vertical section on line I—I of Figs. 326 and 328.

the combustion chamber by means of the ports *c*, which alternate with the gas ports *d*. The products of combustion pass downward through the opening *A* into the canal *K*, whence by means of the downtake *L* they are

divided to the right and left, entering the horizonal flues of the recuperative chambers *FF*. These flues are divided into two series, so that the products of combustion are caused to make a return pass, as shown in Fig. 325, finally entering the flue *M*, from which they are led to the chimney.

In the patent specification the inventors of this furnace state that in practice (presumably at works of the New Jersey Zinc Co.) they have ascertained that its operation "is accompanied with a notable increase in the

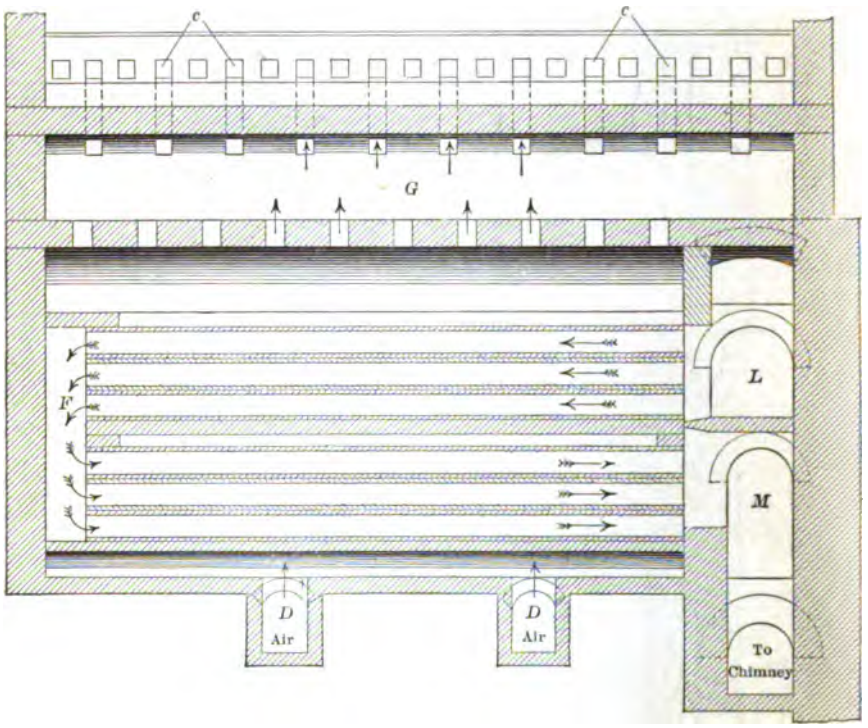
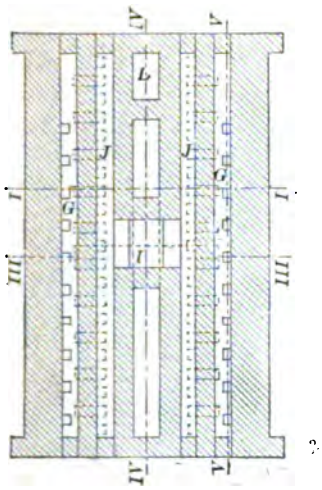


FIG. 325. CONVERS & DE SAULLES FURNACE.
Longitudinal section through recuperative chamber on line V—V of Fig. 327.

output of spelter, a considerable saving in fuel, and a marked decrease in the loss of retorts incident to the usual forms of Belgian furnaces." It will be observed that the gas entering from the producers is conveyed through the canals *JJ*, situated between the recuperative chambers and the flues for the outgoing products of combustion before the latter have entered the recuperative chambers, in consequence whereof the gas itself is preheated to a very high degree, it being found that the temperature prevailing in the

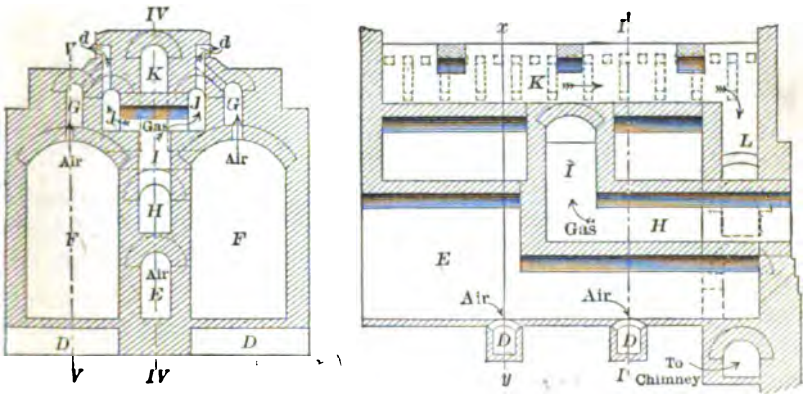
canals *JJ* is so high that their brickwork is raised to cherry red, which is almost if not quite equal to the temperature of the walls of the canals *GG*.



FIGS. 326 TO 328.

CONVERS & DE SAULLES FURNACE.

Fig. 326: Horizontal section on line II–II of Fig. 324. Fig. 327: Vertical section on line III–III of Fig. 326. Fig. 328: Vertical section on line IV–IV of Figs. 326 and 327. Figs. 326 and 328 are engraved on approximately one-half the scale of Figs. 324 and 325. Figs. 327 and 328 show only the portion of the furnace below the combustion chamber.



Natural Gas Fired.

In the United States there are three districts where zinc smelters use natural gas as fuel. These are the vicinity of Marion, Ind., Iola, Kan., and Cherryvale, Kan. The first is a long distance from the mines of Kansas, Missouri, Wisconsin and Tennessee. The second and third are near the mines of Kansas and Missouri. With natural gas available as fuel and of so little pecuniary value as it was held to be in each of the districts above

mentioned, the distillation furnace is reduced to its elemental form, i.e., a simple combustion chamber in which the gas can be burned around the retorts, without fireplace and without heat recuperative system. Natural gas firing is obviously applicable to any type of furnace. In the United States it has been applied to two types, namely, the ordinary Belgian and the Hegeler-Belgian. The first natural gas fired furnaces were built in Indiana.

INDIANA FURNACES.—The Indiana furnaces are essentially simple Belgian furnaces into which the gas is turned under the lowest rows of retorts. The hearth of the combustion chamber corresponds with the ground level and there are no ash tunnels below it. At the Columbia Zinc Works (James Latourette) at Marion there are two furnaces with 100 retorts per side, arranged in five rows of 20 each, making 200 retorts for the furnace. From a gas main above the furnace a branch (¾ in. pipe) is brought down in front of every buckstave or stanchion. From these branches a horizontal bent pipe leads into the furnace on each side of the buckstave, passing through a circular hole in the front walls of the furnace, which hole is large enough to permit air to enter around the pipe; there is one of these branches under the lowest row of retorts and another from the same vertical pipe under the second row from the top. Besides the air which is drawn in around the gas pipes, air is also blown in under a slight pressure from two galvanized iron mains extending longitudinally over the furnace. The alternate stanchions of the furnace fronts are rectangular, hollow iron castings, with openings corresponding to ports through the pillars of the front lining against which the stanchions abut. These stanchions are connected with the air mains and air is blown into the furnace through them. There are also two flues in the middle wall, one each side, with ports opening into the combustion chamber just under the lowest row of shelves. These flues are connected with the air mains by galvanized iron pipes leading down against the end walls of the furnace. The furnace has two chimneys, one at each end. The gases of combustion escape through four flues on each side into canals leading to the chimneys. These furnaces were built in 1897.

The furnaces built originally at Marion are single combustion chambers, each containing 400 retorts arranged in five rows of 80 each. There are two furnaces in a house, set back to back with a wide alley between them. The rear walls are 9 in. thick, not including the lining. The gas is introduced at 8 oz. pressure through small pipes, fitted with air mixers on the Bunsen burner principle, passing through holes (13 in. apart) in the back wall under the lowest row of retorts. The products of combustion escape through four flues in the roof into canals communicating with four chimneys, each chimney being common to the two furnaces. The furnaces are

FIG. 329. DISTILLATION FURNACE IN OPERATION AT INGALLS, IND.

divided by parting walls into sections, worked by independent crews, and are therefore a series of simple furnaces built end to end. The firing is under the control of one man.

At Ingalls, Ind., there is one furnace of $2\times4\times40=320$ and one of $2\times4\times53=424$. The retorts at Ingalls are 48·5 and 51 in. long and 8 in. diameter inside. The gas is admitted under the lowest row of retorts through pipes inserted between each tier of retorts, and between the second and third rows by a pipe at every other buckstave. Air is admitted freely around the lower gas jets. Each side of the 320 retort furnace has 10 flues, which communicate with a common flue built on top of the middle, longitudinal wall. The accompanying engraving is from a photograph of this furnace in operation. At Upland, Ind., there are double furnaces with $2\times5\times22=220$ retorts and $2\times5\times26=260$.

The Edgar Zinc Co. at its works at Cherryvale, Kan., adopted a furnace of the Indiana type, but of more economical design and more substantial construction. It has 200 retorts, 100 on each side arranged in five rows of 20 each. There is a central tunnel under the furnace on the arch of which the middle wall stands. Through the arch on each side there are ports leading into the combustion chambers, under the lowest row of retorts, through which the gas pipes are inserted, the gas being taken from a main in the tunnel. Along the two fronts of the furnace there are horizontal gas pipes, just below the third row of retorts (counting from the lowest upward) from which gas is admitted into the combustion chambers between every two retorts, i.e., at each stanchion. The gas enters the furnace at 4 oz. pressure. There are three furnaces in a house, set end to end but with considerable space between them, an arrangement which permits of a satisfactory arrangement of service tramways and makes the furnace house a reasonably comfortable place in which to work. The general arrangement of the Cherryvale works is shown in the engraving on page 612.

Iola Furnaces.—The natural gas fired furnaces employed in the vicinity of Iola, Kan., are direct adaptations of the Hegeler furnace, with which the original Iola type was almost identical; but the difference between the properties of natural gas and those of coal gas compelled the introduction of modifications in order to obtain satisfactory results. Most of the furnaces in this district have 600 retorts=300 per side, arranged in five rows of 60 each, or 620 retorts=$2\times5\times62$, and $660=2\times5\times66$. The retorts are cylindrical, 50 in. long outside and 8 in. in diameter inside. In the earliest furnaces the gas was admitted by means of pipes, fitted with Bunsen burners, passing through ports in one of the end walls, and traversing longitudinally through the combustion chamber the burned products escaped through chimneys at

the other end. Air was also admitted at intervals through the long sides of the furnace. Great trouble was experienced at first by the deposition of soot around the retorts from the dissociation of the hydrocarbons of the gas, especially at the gas inlet end. These deposits would frequently choke the spaces between the retorts and would have to be barred out. In the later furnaces the gas was introduced, together with the air, at intervals along the front of the furnace; or it was introduced partly in that manner and partly through one end as was done originally. In the latter case the first few feet of the combustion chamber was free from retorts, affording the gas an opportunity to mix thoroughly with air and be in active combustion before enveloping the first tier of retorts. In the front wall of the mixing chamber there were doors, generally bricked up, which could be opened for the

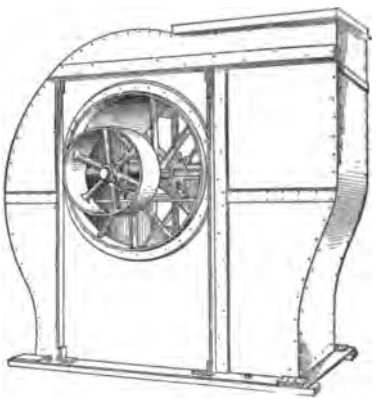


FIG. 330. TYPE OF FAN BLOWER USED AT IOLA, KAN.

removal of soot. In the standard furnace of the present time the gas and air are introduced only through the fronts, and the mixing chamber has been abandoned as unnecessary.

The gas is admitted to the furnaces at a pressure of 4 to 6 oz. In the early practice higher pressures, ranging from 8 to 16 oz., were employed, but subsequent experience demonstrated the advantage of lower pressures. The diminution in pressure from that which is carried in the mains of the works and its control are effected by means of Chaplin-Fulton regulators such as are described in Chapter VIII. A regulator of appropriate size is installed at each distillation furnace, placed originally in close proximity therewith, but the explosion of one of them, leading to the destruction by fire of the distillation furnace and its house, has caused the regulators to be set back about 50 ft. from the furnace house, a small shed protecting

each regulator. Each furnace should have an ordinary mercury manometer for determination of the gas pressure, and a water gauge for the air.

The blowers which furnish the air supply are installed in the engine house, whence the air is conveyed to the furnaces through galvanized iron mains and branches, which are supported by light trestles. The fan-blowers manufactured by the Garden City Fan Co., of Chicago, Ill., are commonly used in the Iola district. It is customary to make the installation in duplicate, so that the operation of the works will not be checked by a break-down. The two fans and their respective driving engines are generally used alternately, night and day. The air pressure at the furnaces is maintained at about 2·2 to 2·8 in. of water, this as well as the gas pressure varying according to the requirements of the furnace at different periods of the distillation.

The dimensions and other data of the sizes of fans used at Iola are given in the following table:

Size	List Price	Average Speed	Width and Height of Outlet	Size of Pulley	Diam. Inlet or Outlet in Round Pipes	Weight in pounds	H. P. required
90″	$300·00	510	28 × 40	20 × 7″	41″	2,050	18
100″	390·00	467	34 × 42	22 × 7″	45″	2,350	25
110″	490·00	425	36 × 48	24 × 8″	48″	2,600	33
120″	575·00	390	42 × 48	24 × 10″	53″	3,000	42

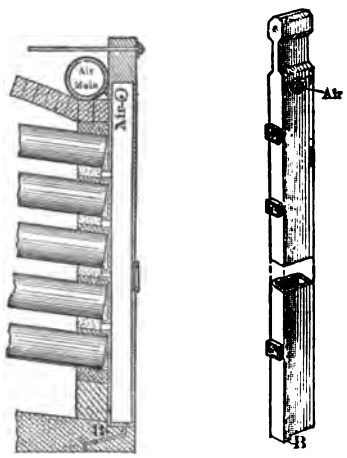
These fans are speeded to give a pressure of 2 oz. (3·5 in. of water) at the outlet, which pressure is of course diminished toward the furnaces according to the length and diameter of the pipes, the number and character of the bends, etc. A 120 in. fan is more than sufficient for a plant of five blocks of furnaces.

Extending longitudinally over each furnace there is a sheet iron air main, commonly 24 in. diameter, from which branches extend to each side of the furnace and communicate with the vertical supply pipes. At many of the works at Iola, including those of the Lanyon Zinc Co., the air is discharged into the combustion chambers through rectangular, hollow cast iron buck-staves or stanchions,[1] precisely as is done at Marion, Ind., which system is the subject of a patent granted to Robert H. and William Lanyon, of Iola. According to their design, which is shown in Fig. 331, the front of the furnace is built up of clay pillars, with clay plates resting horizontally upon them, in such manner as to divide the front of the furnace into a series of niches, each of which accommodates two retorts. The plates resting on every alternate pillar abut closely, while a space is left between the meeting

[1] U. S. patent, No. 616,475, Dec. 27, 1898.

ends on the intermediate pillars. This results in the formation of a series of port holes into the furnace, after the front has been closed by the retorts which are luted up in the ordinary manner. Corresponding to each pillar there is a buckstaff, the alternate ones being made hollow, with rectangular openings or thimbles on one side, which fit into the ports in the furnace. At their upper ends the hollow buckstaves are connected with the main air supply pipe.

Hollow buckstaves like those described above are used in connection with nearly all of the Iola furnaces, but there are unimportant modifications in details. Sometimes the air is brought down through a 4 in. wrought iron pipe, which may stand against a recessed cast iron buckstave or between a



FIGS. 331 AND 332. LANYON HOLLOW BUCKSTAVE.

pair of ordinary, rectangular cast iron buckstaves. Sometimes the gas pipes are introduced into the furnace at one side of the air pipe and sometimes they pass directly through the air pipe; in any case it is important to provide means by which a bar can be inserted to clear the gas and air port of any obstructions that may form in them. The fire clay pillars and shelves are also subject to slight modifications to correspond with the various arrangements of the gas and air pipes.

A furnace built at Iola in 1899 is illustrated in the accompanying engravings from photographs. One shows the appearance of the furnace from the outside and the other the appearance of one of the combustion chambers looking from the chimney end to the gas inlet end; the retorts had not been inserted. The total length of the furnace outside is 79 ft. 8 in. The inlet

FIG. 333. EXTERIOR OF A DISTILLATION FURNACE AT IOLA, KAN.

The photograph shows the furnace just completed and ready for the retorts, of which there are 310 per side.

end wall is 23 in. thick (two and a half brick); the outlet end wall is 18·5 in. (two brick); the combustion chamber is consequently 76 ft. 2·5 in. long. There are 310 retorts per side (five rows of 62 each). There being two retorts in a niche, each row is divided into 31 niches, besides which there are two niches next the gas inlet end in which there are no retorts; the latter niches are ordinarily bricked up and thus enclose the mixing chamber. As will be seen from the engraving of the interior of the furnace there are low bridge walls rising from the hearth and wings extending downward from the roof arch. Going from the chimney end and counting the first half pillar of the front wall as No. 1, there are bridge walls on the hearth at Nos. 6, 10, 14, 18, 22, 26 and 30, while there are wings from the arch just beyond Nos. 4, 6 and 8. These are intended to throw the flames up or down to the retorts. The bridges are 9 in. wide, 11 in. high at the end next to the middle longitudinal wall of the furnace and 6 in. next to the front walls. Distortion of the furnace is provided against by leaving open slots 2 in. wide cutting transversely through the roof arch and middle wall. There is such a slot at pillar No. 12 and another at No. 22. Until these close up they are covered by loose bricks laid on the arch. The expansion of the front wall is allowed for by some loose joints. It will also be observed from the photograph that the end walls are set in clear of the side walls and arch. The older Iola furnaces in which proper provision against expansion was not made become badly canted at the ends after a short time under fire.

The chimneys, of which there are two, one for each side of the furnace, are 50 ft. high above the floor of the furnace. They are rectangular in cross section, 31×36 in. inside, with walls 9 in. thick. The air main over the furnace is 24 in. in diameter. There are branches at every other buckstave, i.e., between every four retorts of a row. These buckstaves are special castings, recessed to receive the air pipe, which is in that portion an ordinary 3 in. wrought iron pipe. There are ports through the pipe, buckstave and front lining of the furnace, as in the Lanyon furnace, but instead of laying the front shelves in such a way as to leave openings, the pillars are molded with holes, the five sections of the pillars forming a continuous column. This construction is not so strong as where the shelves and pillars are bonded together, but it makes it much more easy to replace a broken shelf. The pillars which are thus tapped for air ports are necessarily wider than those which are not. A 3 in. wrought iron gas main is run along each side of the furnace, overhead, and small pipes lead down from it beside the air pipes with branches passing into the ports in the front lining; these branches do not go through the buckstaves, but pass to one side of them. In the air pipes there are holes, into which plugs are screwed, corresponding with the

ports into the furnace so that a bar may be thrust through to remove any obstruction. The pressure of the gas is reduced at the wells and again at the furnace by the ordinary form of natural gas pressure regulator. There is a regulator for each furnace which is under the control of the brigadier or fireman, as he is called at Iola.

Recent Modifications.—Even since the writing of the foregoing descriptions, or since the end of 1899, important changes have been made in the equipment of the distillation furnaces at Iola, where the practice has in the meanwhile passed beyond the experimental stage and has now become well crystallized. The gas is now introduced into the furnaces at a considerably lower pressure than it was originally, which has necessitated a readjustment of the supply pipes. Most of the furnaces of 1899 have been entirely repiped. Gas is no longer admitted through the end wall, the whole supply being introduced through the lateral pipes. The mixing chamber has been discarded as unnecessary and its room is now occupied by retorts, the number of which is thereby increased 20 per side, or 40 per furnace. The masonry of the furnaces is built more substantially and the iron work is better done, giving a longer campaign.

In the latest furnaces the gas, reduced by the regulator to the proper pressure, passes to the furnace through a 4 in. wrought iron pipe, from which a branch, also of 4 in. pipe, is conducted along each side of the furnace, supported by brackets from the buckstaves above the line of the highest row of retorts. From the side mains, vertical branches of 1 in. pipe are led down in front of each alternate buckstave, and from those branches two feeders, each of 0·5 in. pipe, enter the furnace, one opposite the second row of shelves and the other opposite the fourth row. The holes in the pillars through which the feeders pass are 2 in. in diameter.

The gas mains on each side of the furnace used to be only of 3 in. diameter; the branches were 1¼ in. and the feeds, of which there were four, were ⅜ in.

The air branches are made of 4 in. wrought iron pipe, which are connected with the main by 4·5 in. galvanized iron pipes. Each branch has a gate, operated by a lever, for control of the air supply. In the gas pipe system there is a valve in each side main, placed conveniently in that part of the pipe which passes by the end wall, and also a valve in each vertical branch.

The chimneys are no longer built integrally with the furnace, but are set a few feet beyond the end wall and a little to one side of the projection of the middle line of each combustion chamber, which permits them to stand on a solid foundation, instead of on the tunnel arch. If the furnaces are

FIG. 334. INTERIOR OF A DISTILLATION FURNACE AT IOLA, KAN.

View looking from the chimney end. This is the same furnace of which the exterior is shown in Fig. 333.

built with two tunnels, as at La Harpe, the chimneys are of course put between them, but generally at Iola the furnaces have only one tunnel, which

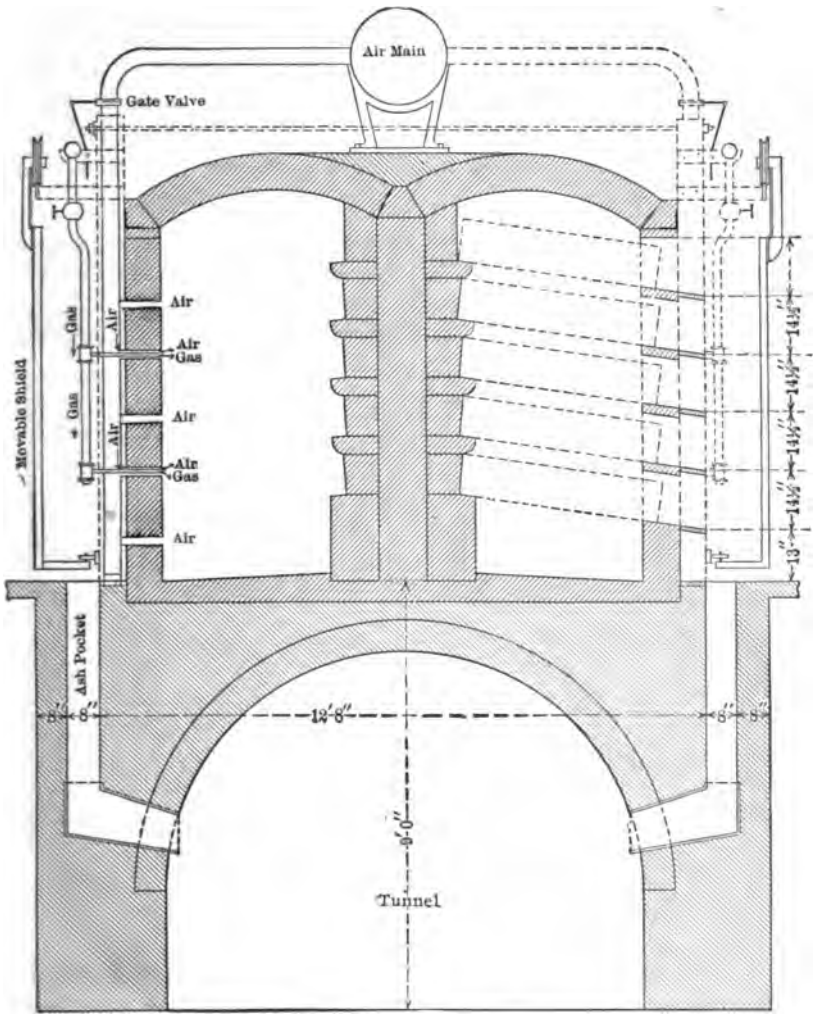


FIG. 335. IOLA FURNACE.

Transverse vertical sections.

Above the floor line the left hand side is a section through the pillars containing the gas and air ports. The right hand side is a section midway between pillars.

is light, airy and cool. The ashes from the retorts are dropped into pockets in front of the furnace, from which chutes project into the tunnels. There

is an ash pocket for each section of the furnace, i.e., 20 retorts. The ashes are drawn from the pockets into a car, which is pulled out by a horse, the floor of the tunnel being below the ground level.

The result of the improvements in the Iola practice have been to increase the length of furnace campaign, diminishing the expense for repairs and renewals; and what is more important, they insure a more perfect combustion of the gas and the ability to maintain an equable temperature in all parts of the furnace. Trouble from the deposition of soot is a thing of

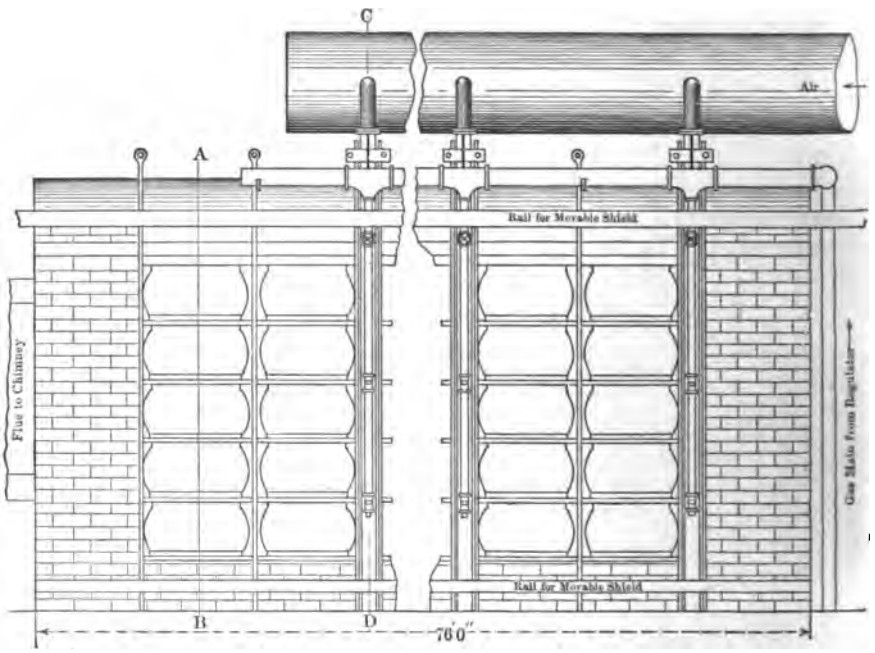


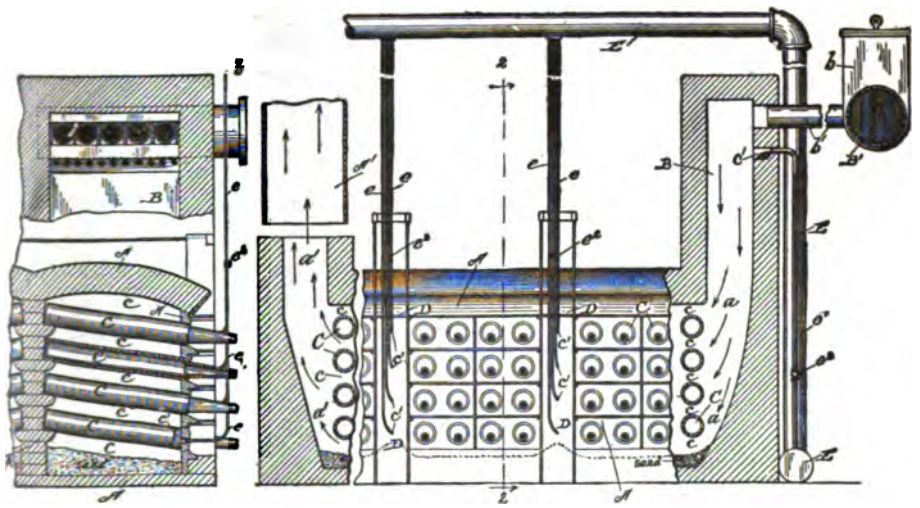
FIG. 336. IOLA FURNACE.
Front elevation, retorts removed.

the past, and the extraction of metal, which was up to 1900 inferior to what was attained by the coal smelters, has now become at least equal to their average work in the best practice, if not somewhat superior.

It appears likely that the "blow furnace," as this type is commonly designated at Iola, with about 600 retorts will continue to be the characteristic Iola type, and the smelters of that district deserve much credit for the manner in which they have developed it, although the idea was not original with them. The chimney draught furnaces, which have been tried there,

have all been discarded as inferior. An interesting innovation, which has lately been made by the Lanyon Zinc Co., has been the erection of some furnaces of the standard Iola type but with only 400 retorts (2×5×40). By the end of 1901 that company will have four such furnaces in operation.

Hegeler Furnace for Natural Gas.—An entirely different form of furnace for burning gas, artificial or natural, was patented by E. C. Hegeler in 1898. In this furnace, which has not yet come into use, the air is admitted at one end and the gas through pipes inserted in the front of the furnace at proper intervals toward the chimney end. This is a reversal of the conditions where the gas enters the furnace at the end, and the air through ports at intervals in its faces. At one end of the combustion chamber there is a



FIGS. 337 AND 338. HEGELER FURNACE FOR NATURAL GAS FIRING.
Fig. 337: Transverse section. Fig. 338: Longitudinal section and front elevation.

flue through which the entire quantity of air to be consumed in the furnace is admitted, enough gas being introduced and burned in this flue to raise the temperature to the required height at the first group of retorts. Further on in the combustion chamber, at different points, enough additional gas is admitted to produce the required temperature in all parts. The admission of gas is regulated so as to consume all the oxygen admitted at the end of the furnace, an excess of gas making itself known by the appearance of flames at the exit end of the combustion chamber.

The construction of this furnace is shown clearly in the accompanying drawings. *B* is the air flue, the portion of it which immediately joins the

combustion chamber being made of brickwork, and the other portion *B'* being made of sheet iron and provided with a regulating valve *b*, the sheet iron portion being preferably connected with the brick portion by means of a series of small sheet iron pipes *b'* entering, through separate openings in the brickwork. The retorts in the combustion chamber are arranged in groups with open spaces at *D* between them. *E* is a large gas pipe, conveying the supply of gas under suitable pressure. This pipe is connected with the combustion chamber by means of small pipes *e*, which enter through holes *c'* in the front walls of the combustion chamber into the open spaces at *D*; *e'e'* are similar small pipes (but somewhat larger than *ee*) which enter the brick portion of the air flue. Each of the small pipes *e* and *e'* is provided with a valve by which the quantity of gas passing through may be regulated. The openings *c'* through the wall of the air flue and the front of the combustion chamber are made somewhat larger than the outside of the gas pipe, the latter being made tight by luting with clay. The gas pipes are sufficiently flexible to be pulled out of the openings and turned to one side for looking into the furnace. The cross sectional area of the fire brick part of the air flue should not be much less than the cross sectional area of all the open spaces between the retorts. Only the requisite amount of gas is admitted into the combustion chamber through the air flue and the sections *DD*, etc., to heat the group of retorts immediately in advance, so that free oxygen may be met by the gas entering the chamber at each section. In this manner it is claimed that a uniform temperature can be maintained throughout the chamber, and the lowermost retorts are heated as easily as the upper. The air, which may be advantageously preheated, is forced through the flue by means of a fan.[1]

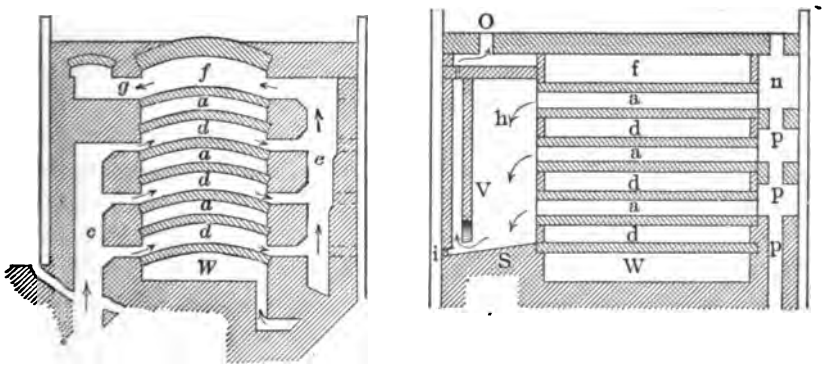
Miscellaneous Furnaces—New Proposals.

The limitations that are imposed on the standard types of furnaces, especially the necessity of performing the distillation in comparatively small vessels and condensing the distilled metal in a comparatively imperfect manner, together with the requirement of ore low in iron and lead, if the best results are to be obtained, have led to numerous attempts to improve the apparatus, aside from the attempts to conduct the process in shaft furnaces, which will be referred to in a subsequent chapter. Although none of these attempts has been successful, it is worth while to describe some of the more promising as a guide to future work. The direction in which improvement appears most promising is in the line of better condensation of the

[1] United States patent, No. 612,104, Oct. 11, 1898.

metal. In so far as the distillation is concerned there does not seem to be much chance that we can abandon the small retorts that we have been restricted to for a century.

FRANCISCI FURNACE.—This furnace, designed by Doctor Victor Steger for the treatment of roasted blende and other fine ore in Upper Silesia and patented in the name of his partner, Herr Francisci of Schweidnitz, consists of a series of superimposed muffles, formed by arches of magnesia brick built into the furnace walls and communicating with a common condensation chamber. The arrangement is shown in the accompanying engravings. Referring to the transverse section, *aaa* are the muffles. The gas coming through the flue *c* burns, with air from *W*, in the chambers *ddd* between the muffles. The products of combustion escape through the flues *e*, *f* and *g*. Referring to the longitudinal section, the muffles are charged



FIGS. 339 AND 340. FRANCISCI FURNACE.

Fig. 339: Transverse vertical section. Fig. 340: Longitudinal vertical section.

from the right hand ends, which are closed during the distillation. The residues are discharged from the same ends, being dropped through the chutes *ppp*. The zinc vapor passes into the common condensation chamber *h*, to the sloping bottom of which it is deflected by the wall *V*. The condensed zinc collected on the bottom is tapped at *i*. The uncondensed vapor, chiefly carbon monoxide, escapes to *O*, where it may be collected for use as fuel.[1]

Steger claims that magnesia is a highly desirable material for muffles, inasmuch as it is 2·5 to 3 times a better conductor of heat than clay, is impenetrable to zinc vapor, is capable of withstanding the highest temperature,

[1] German patent, 76,285; Belgian patent, 107,606; English patent, 23,070; Austrian patent, 44-3256; United States patent, 526,-808; Steger, "Verdichtung der Metalldämpfe in Zinkhütten," Sammlung chemischer und chemisch-technischer Vorträge, I, ii, 67.

and is not corroded by the gangue of the ore. The advantages that would be derived from the use of such large muffles are obvious. Even more obvious is the advantage of a common condensation chamber, in which the metallic vapor can be drawn over a bath of molten metal, promoting the condensation of the metallic particles in the vapor by adhesion and cohesion, while the temperature of the condensing chamber can be carefully controlled by means of air cooling flues in its walls. Experiments have been made with the Francisci furnace in Upper Silesia, but so far nothing is known by me as to the result except that the magnesia brick used at first were said to be very susceptible to change of temperature.

LYNEN FURNACE.—Leo Lynen has patented a furnace with a common condensation chamber, which is illustrated in the accompanying engravings. The latter show a massive of Rhenish furnaces with two condensing chambers. Referring to the engravings, it will be seen that there are shown four distillation chambers, each containing 36 retorts. Each pair of combustion chambers is separated by a narrow chamber, parallel with and of the same length as the combustion chambers which it separates. The retorts are open at both ends. The outer ends rest on shelves in the ordinary manner. The inner ends also rest on shelves, but they are set so as to overlap or telescope with short tubes fixed in the inner wall, there being one of these tubes for each retort. When the retort is in position the joint between it and the tube is luted tightly with clay from the inside of the retort. The outer end of the retort is closed by a tightly luted clay plate, which is removed during charging and discharging. The retort has thus a constant communication with the condensing chamber between the distillation chambers. The bottom of the condensing chamber is made of trough shape with a slope toward the end of the furnace. There is a tap hole in the wall through which the metal can be drawn off. The carbon monoxide gas is led off through a flue above the bath of metal, and of course can be conducted through any kind of an auxiliary condensation system that is desirable. Inside of the condensation chambers there are pipes through which air may be circulated, thus giving control over the temperature in the condenser.

It is claimed by the inventor that the possibility of maintaining the temperature at the exact point that is required, the condensation of the metallic vapor in the presence of a large bath of molten metal and the exhaustion of the uncondensed gas through a few mains will be highly advantageous, and there can be no doubt that he is correct in that assumption. If the system will work as contemplated it ought to be possible to effect a complete condensation of the zinc that is volatilized, save what is absorbed by

and filters through the retorts and escapes through cracks in the latter. Doubtless he is correct also in the claim that with lead bearing ore a metal in which there will be concentrated a large part of the lead that has been

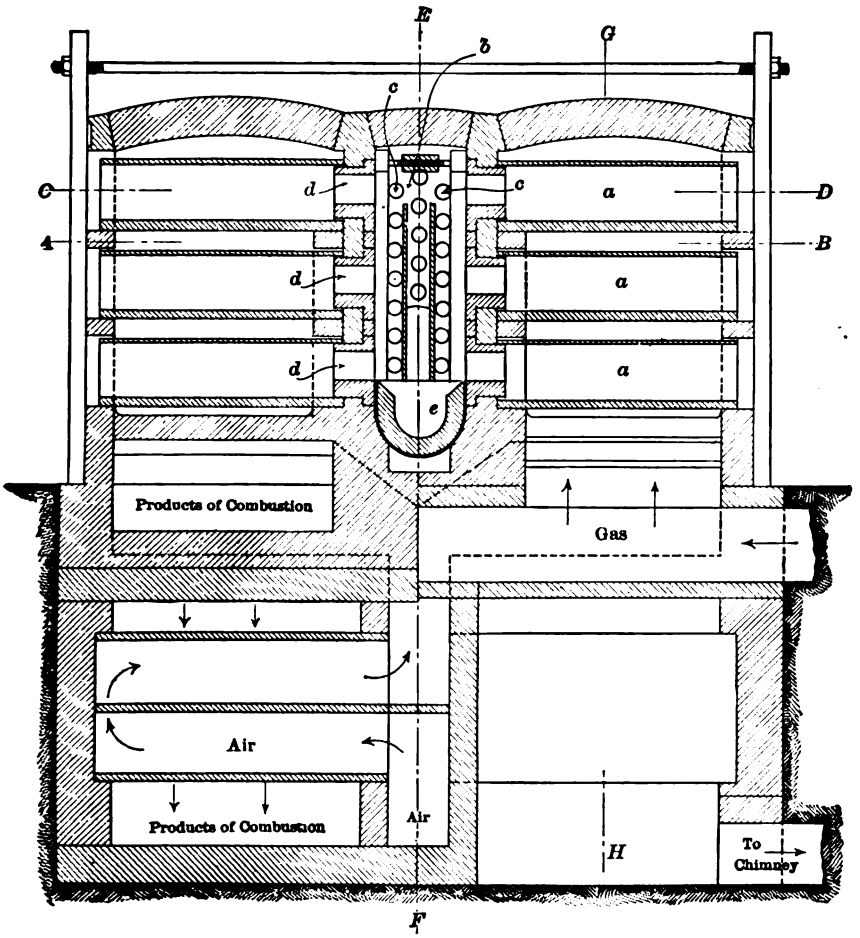


FIG. 341. LYNEN FURNACE.

Transverse vertical sections. The left hand side is a section on the line PQ of Fig. 342; the right hand side is a section on the line IKLMNO.

volatilized and condensed can be drawn off from the bottom of the bath and a spelter low in lead from the top, the condenser playing the part of a refining furnace; moreover the possibility of handling the metal without

much use of iron tools would tend to give a product low in iron. The inventor claims further advantages in the possibility of exhausting the last of the vapor from the retorts and in reducing the time of distillation by shortening the maneuver, but these advantages are not so clear. As to shortening the maneuver so that the distillation may be conducted in less than 24 hours it is doubtful. It is true that labor would be saved, there being no condensers to be removed, cleaned and replaced; the cost of the condensers would also be saved; but the handling of the condensers goes on usually simultaneously with the discharging and charging of the retorts, or nearly so, and it does not seem as if much time would be saved so long as the retorts must still be charged by hand.

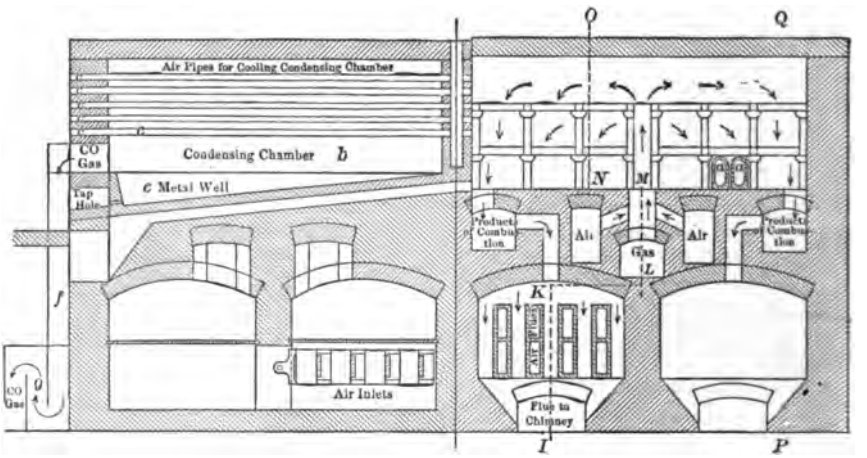


FIG. 342. LYNEN FURNACE.

Longitudinal sections on lines EF and GH of Fig. 341. The left hand side is a section on the line EF, and the right hand side on GH.

The chief difficulty that can be foreseen with the Lynen furnace as at present designed is the making of a tight joint between each retort and the inner wall, and the danger of leakage thereat. This has been recognized by Mr. Lynen himself and recently he has made some improvement intended to obviate it. Since then, experiments with the furnace have been made at a Rhenish works, but their results have not been communicated. The idea embodied in this furnace undoubtedly has merit and may be developed into something useful.

The Lynen system of condensation is obviously not restricted to the type of furnace which he has shown, though in occupying the place of the solid

middle wall, or the place where it would otherwise be, the ordinary design of Belgian and Rhenish furnaces must necessarily be modified. In the accompanying engravings the furnace shown is of the counter current recuperative type, but the air and gas flues are set at right angles to the long axis of the furnace instead of parallel with it as shown in Fig. 282. The gas comes in through four canals in the long sides of the furnace, one

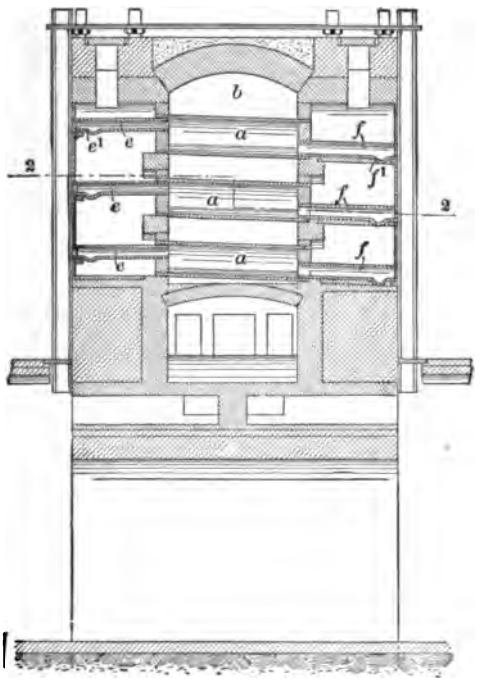
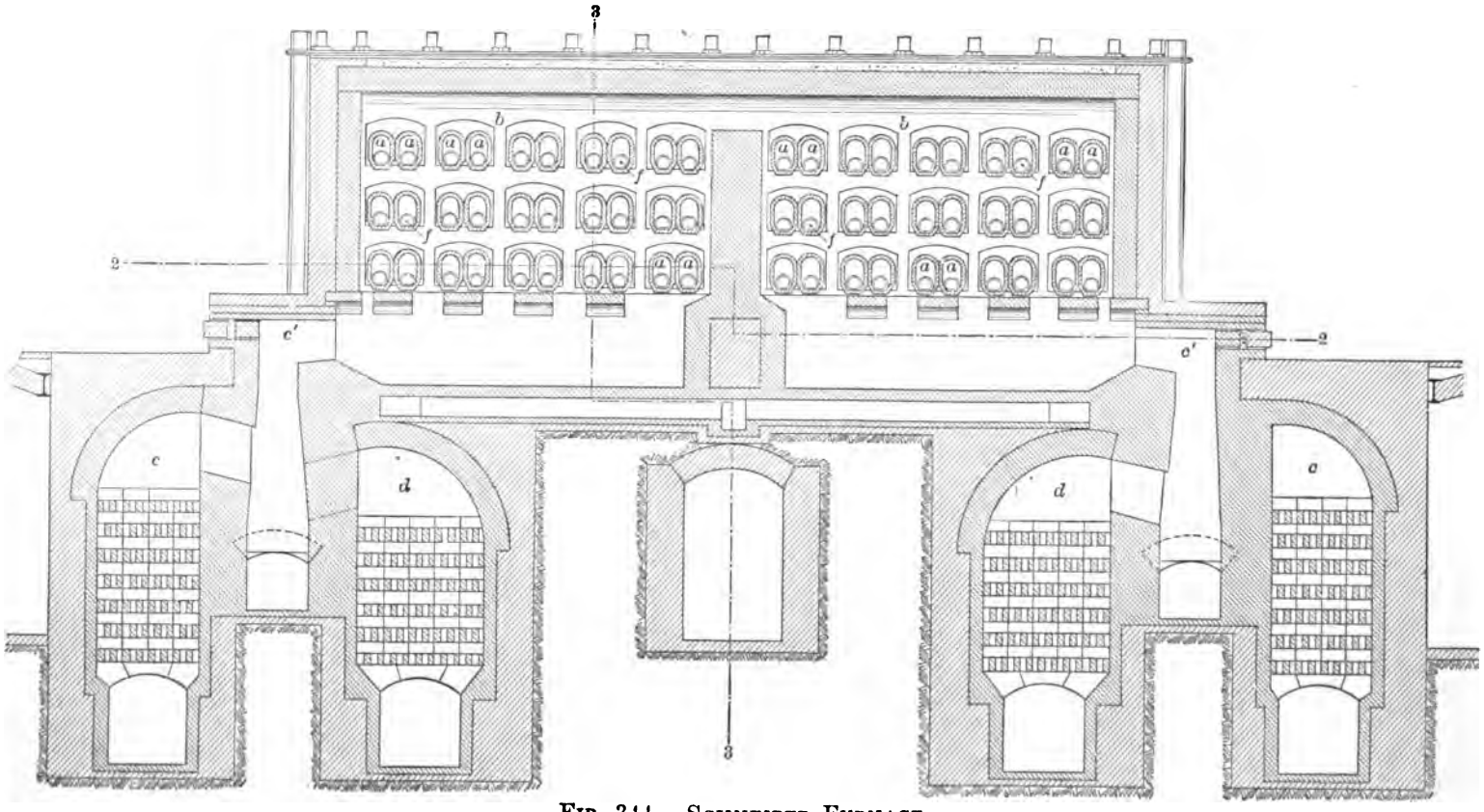


FIG. 343. SCHNEIDER FURNACE.
Transverse vertical section.

canal for each combustion chamber. Rising through central flues in the hearths it mixes with preheated air and the products of combustion circulate as shown by the arrows in one of the longitudinal sections, escaping through the flues as shown and enveloping the air preheating flues on their way to the chimney. The air enters through longitudinal flues in the middle of the furnace, these flues being under the condensing chambers, and enters the recuperative flues through the ports on each side, which are equipped with dampers for regulating the supply. From the recuperative



FIR. 344. SCHNEIDER FURNACE.
Longitudinal vertical section.

flues it passes through canals leading to the stream of gas. Lynen's patent covers the system of condensation and not any particular type of furnace.[1]

SCHNEIDER FURNACE.—Richard Schneider, of Dresden, Germany, patented a distillation furnace for the reduction of zinc ores containing lead,

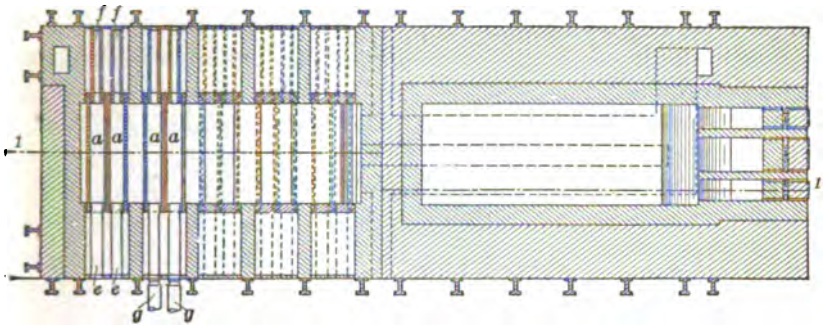
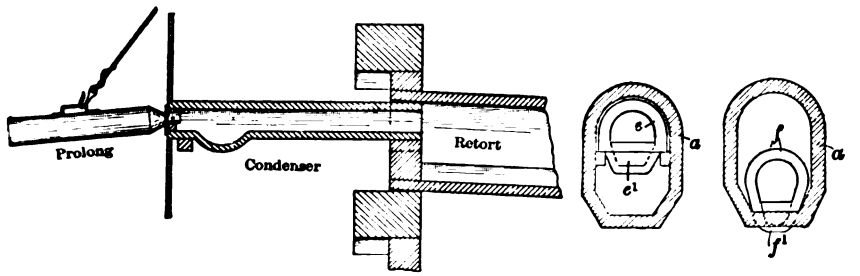


FIG. 345. SCHNEIDER FURNACE.
Horizontal section on line 2—2 of Figs. 343 and 344.

and the recovery of the latter metal as well as the zinc. This furnace consists essentially of a single combustion chamber in which are placed three rows of retorts, the latter being clay cylinders open at each end, placed in



FIGS. 346 TO 348. SCHNEIDER FURNACE.
Fig. 346: Connections of retort, condenser and prolong. Figs. 347 and 348: Cross-sections of retorts and condensers, in which *a* is the retort, *e* the zinc condenser, *f* the lead collector, while *e'* and *f'* are sumps in the condenser and collector respectively.

the furnace at a suitable inclination. A condenser is connected with the higher end in the ordinary manner. A similar condenser is connected with the lower end, but opens into the lower half of the retort instead of into the upper half as in the case of the ordinary condenser. The idea is that the zinc will be volatilized and condensed in the usual manner, while the lead reduced will trickle down the incline and collect in the condenser at the

[1] British patent, No. 16,180, Aug. 28, 1893.

lower end.[1] There is no novelty in this idea. George M. Hölstein, of Pulaski, Va., has designed a special retort for the same purpose. This retort is placed in the furnace in the ordinary manner. The clay plate which closes the end under the condenser is provided with a tap-hole, through which any lead accumulated in the retort can be drawn out.[2] The drawings of Schneider's furnace are interesting as showing a project for heating the retorts by a longitudinal movement of the gases, alternately in opposite directions as in the Ferraris furnace, but with a pair of regenerators at each

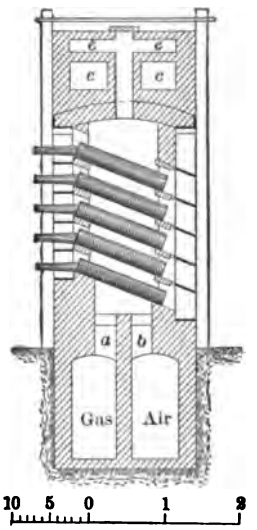


FIG. 349. THUM FURNACE.

end as in open hearth steel furnaces, though in this case the current is deflected over a vertical wall, while an open space is left under the ranks of retorts on each side, both of which arrangements would be unnecessary if a little free space were left at each end of the combustion chamber, as has been proved by the experience with the Ferraris furnace.

THUM FURNACE.—F. A. Thum designed a gas fired furnace especially for the treatment of zinc ore which is high in lead. There was a single combustion chamber, as shown in Fig. 349, with five rows of retorts set with a comparatively great inclination. The retorts were simply tubes open at each end. The higher ends were fitted with condensers in which the zinc was collected in the usual manner. The lower ends were closed by clay plates tightly luted in. The plates were provided with tap holes

[1] United States patent, No. 605,802, June 14, 1898.
[2] United States patent, No. 554,185, Feb. 14, 1896.

through which the reduced lead accumulating at the lower end could be drawn off. The gas and air for heating the retorts were introduced through the canals in the base of the furnace, issuing through the ports *a* and *b*. The products of combustion escaped through the flue in the roof into the canals *ee*, while the air for combustion, before reaching the main canal in the bottom of the furnace, is preheated in the flues *cc*.[1]

So far as I can discover, there is no record in metallurgical literature that Thum's furnace was ever tried practically, i.e., not as designed for the simultaneous recovery of zinc and lead. The form of furnace shown in the accompanying engraving had been employed previously, however, by Thum at Sunderland, Great Britain, for the heating of Belgian retorts arranged in the ordinary manner, with entirely satisfactory results. Thum thought that his proposed retorts, open at both ends, besides enabling ores high in lead to be smelted profitably, would have the further advantage that the condensers would not need to be removed after the distillation of each charge, and would therefore last longer and incidentally thereby reduce the small loss of zinc which is suffered by absorption in the clay of the condensers.

Charging Cars, Furnace Shields and Tapping Ladles.

The introduction of long distillation furnaces, comprising a large number of retorts operated by a single crew of men, has made it feasible to apply certain mechanical devices, which would be too costly in view of the saving they would effect if used with the small direct fired furnaces. The latter are always provided with a suspended shield for each side, which can be pulled down in front of the furnace to protect the men during the maneuver, and generally have a crane for the purpose of handling the pots of molten metal. With the long furnaces, however, it is economical to lay two parallel lines of tramways in front of each furnace and operate thereon movable apparatus for working the furnace. Such apparatus comprises charging cars, retort cleaning cars and metal drawing ladles, independent or combined with movable shields.

Charging Cars.—The charge of ore and coal mixed mechanically in a separate department is loaded in cars for transport to the furnace. These cars are arranged with a gate in one end, at the bottom, through which the mixture can be removed by means of a charging scoop and thrown directly into the retorts. When the furnace is only five retorts high all of them can be filled by a man standing on the floor. At Iola cars of a size that will

[1] Berg.-u. Hüttenm. Ztg., 1875, p. 1.

hold one fourth or one sixth of the charge of a 600 retort furnace, or approximately 6000 or 4000 lb. of ore respectively, plus the proper proportion of reduction material, are employed. The total number of cars which must be provided is therefore four or six times the number of furnaces, the charge for each day being mixed and put in the cars on the previous day, so as to be ready when wanted.

METAL DRAWING CARS.—At Lasalle, Ill., and Laharpe, Kan., the metal is drawn from the condensers into a large pot hanging from a crane which is fixed upon a four-wheel truck. The pot is raised and lowered by a winch. When full of metal it is swung out from the furnace and its contents poured into molds standing upon a truck on a parallel track. The metal-drawing car is commonly provided with a shield, which protects the operator from the heat radiated from the furnace.

CLEANING CARS.—At Lasalle, Ill., the residuum is raked out of the retorts with the aid of a car, on the platform of which the operators stand, On the side next the furnace there is a shield of sheet iron slats, like a window blind, to protect the men. A vertical opening in the shield is provided with grooved rollers, corresponding in position to the four, five or six rows of retorts, as the case may be, which support the heavy iron rabbles employed in the operation.

MOVABLE SHIELDS.—The distillation furnaces at Iola, Kan., are generally provided with movable shields, one per side, which serve both for the functions of metal drawing and cleaning the retorts (the latter operation being commonly performed at Iola by blowing out the ashes with steam).

Chapman Shield.—This shield, which has been patented by Charles S. Chapman of the Pittsburg Foundry and Machine Co., of Pittsburg, Kan., consists essentially of a pair of vertical plates of sheet iron or steel, arranged with a ladle between them, which can be moved longitudinally along the front of the furnace. Each front of the furnace is fitted with a longitudinal track supported by brackets, which are bolted to certain of the buckstaves, above the uppermost row of retorts. On this track run two grooved wheels with hangers which carry the shield. The axle of one of these wheels is fitted with a sprocket to receive a chain for moving the apparatus. The shield consists of two plates 30 in. apart and stiffened by angle iron; each plate is provided with a wing hinged to it, forming extensions of the shield. The hangers are bolted to a horizontal bar *E* and the lower ends of the plates *DD* are held in proper position by means of a bar *F*, which is curved inward toward the furnace in order that it may not interfere with the ladle. Bolted to the buckstaves near the bottom of the furnace and extending the whole length of the latter is a guide rail, having a vertical

outer face, and the plates *DD* of the shield are fitted with brackets carrying rollers, which bear against the guide rail and steady the lower ends of the plates when the shield is moved longitudinally along the furnace. The weight of the shield tends to throw its lower part inward toward the furnace, wherefore the rollers are always in contact with the guide rail. One of the plates of the shield is fitted with a horizontal shaft *I*, carrying a hand-wheel and a sprocket; the latter being connected by a chain with the sprocket of the corresponding grooved wheel running upon the upper rail,

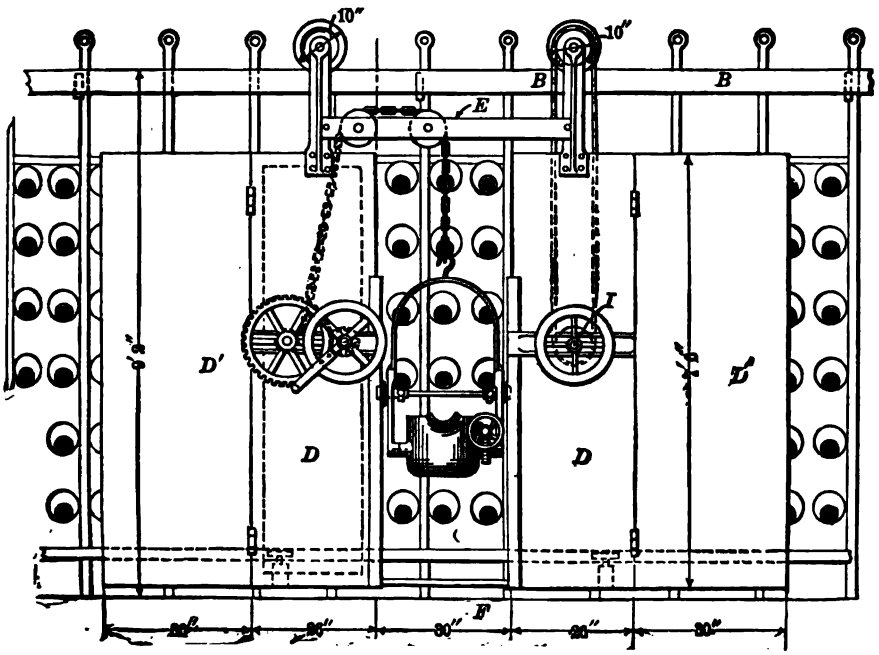
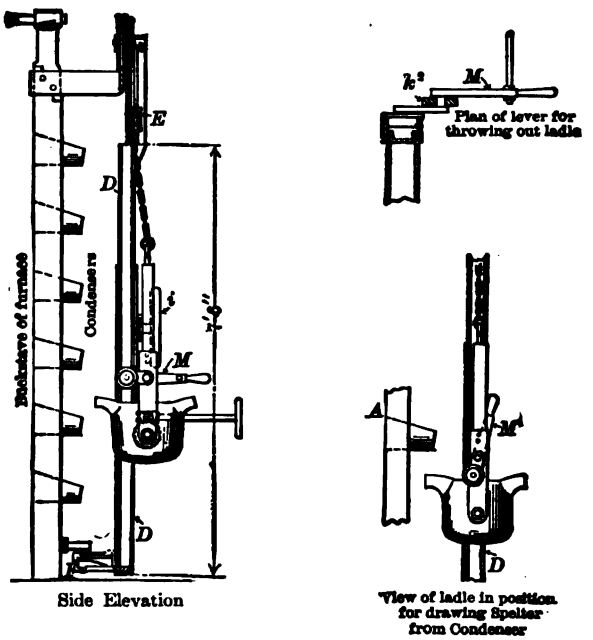


FIG. 350. CHAPMAN FURNACE SHIELD.
Front elevation.

the shield can be moved by means of the hand-wheel in either direction along the furnace.

Between the two main plates of the shield there is arranged a ladle, provided with a spout on its inner and outer sides and trunnions which rest in the lower ends of the two vertical arms. The upper ends of these arms are connected by a bail, from the hook of which the ladle is suspended by a chain. This chain passes over a sheave on the horizontal bar *E*, and thence over another sheave on the same bar down to a windlass which is fixed in

the framework of one of the plates of the shield. By means of this windlass the ladle can be raised and lowered. In order to guide the ladle in its vertical movements there are rollers placed on the ends of a bent lever *M*, which rollers engage in vertical grooves on the inner edge of each plate *D*. The lever *M* is pivoted on the hangers by which the ladle is suspended, forming a toggle joint. By pulling down the lever the ladle is thrust outward, so that it will avoid the ends of the condensers, when it is raised and lowered. One of the trunnions of the kettle is provided with a worm-wheel and the adjacent hanger has a worm-shaft with a hand-wheel at its outer



FIGS. 351 TO 353. CHAPMAN FURNACE SHIELD.

end. By means of this device the ladle can be tilted to pour its contents into a mold or other receptacle. The ladle can be disconnected from the suspending chain and removed so that the shield can be used without it for such work as taking down the condensers and cleaning out the retorts, when the ladle is not needed and would be in the way.

Furnace shields of this type are used in connection with the natural gas fired distillation furnaces which are employed in the Iola district and at Cherryvale, Kan. On account of the great heat to which they are subjected, roller bearings are employed largely in their construction. The

ladle holds about 400 lb. of metal. The shields cost $125 apiece, not including the tracks, f. o. b. Pittsburg, Kan., and weigh approximately 1500 lb. per shield.

Lanyon Shield.—This consists of an iron curtain, stiffened by ribs of angle iron or in any suitable manner, hanging from trolley wheels running on a rail, parallel and on a level with the spring of the arch of the furnace, and just beyond the vertical plane of the outer ends of the condensers. The bottom of the shield is also provided with wheels running on a rail, to prevent swinging. In the shield there are small oval holes, coinciding with the retorts, through which the steam pipe used in Kansas zinc works for the discharge of residues from the retorts may be introduced. Between two sections of the shield there is a vertical opening, properly stiffened by angle iron, with transverse lazy bars, through which rakes, for the discharge of the residues, or other working tools, may be inserted. A kettle which can be raised and lowered and can be swung out from the furnace for pouring into molds is also fitted in the opening to facilitate tapping zinc from the condensers. The shield is moved along in front of the furnace, corresponding with the retorts which are being attended to.[1]

[1] William and Josiah Lanyon, United States patent, No. 621,577, March 21, 1899.

XI.

PRACTICE IN DISTILLATION.

The practice in distillation is essentially the same, whether the furnaces be of the Belgian, Rhenish, or Silesian type, though there are numerous variations in the details; with the tendency toward a common type of furnace, which has previously been referred to, these variations tend also to disappear. For distillation in the large muffles of the Silesian furnace, the calamine ore is not ground so fine as for the small retorts of the Belgian furnace, but blende is necessarily ground fine as a preliminary to roasting and it does not work so well in the large Silesian muffles as in the smaller Rhenish, hence the tendency toward the adoption of the latter in Silesia. With the Silesian furnace, the condensers are not removed during the maneuver; they are now generally arranged to discharge their escaping fumes into a common collecting flue (Dagner and Kleeman condensers). The Rhenish furnaces also are commonly discharged and recharged without removing the condensers, but in connection with muffles (retorts) of the same size used in the United States the condensers have been taken down. The condensers of the Belgian furnace are necessarily removed during the maneuver. Formerly the Belgian furnaces were charged twice per 24 hours and in Upper Silesia the charge remained originally in the retorts for 48 hours, an additional charge being introduced at the end of 24 hours when the distillation was partially completed. In Belgium the first modification of method was to charge the upper rows of retorts only once and the lower ones twice per 24 hours. At present it is the common practice everywhere to distil only one charge per 24 hours.

PUTTING A FURNACE IN OPERATION.—The newly constructed or freshly relined furnace is best left for a few days, the longer the better, to dry naturally. This is not essential, but it is safer than drying artificially, though the latter may be done well enough if it be not hurried. In either case as soon as the furnace is ready for regular firing, the full quota of retorts is put in place and the spaces between them in the niches are closed in and luted up with clay. The retorts do not need to be annealed or

burned; that will be done gradually, as required, in the furnace. The retorts being in place the heating is begun, very gently at first and gradually increasing in intensity until the proper temperature for charging the retorts is attained.

The method of heating the furnace initially depends upon the kind of furnace, i.e., whether it be direct fired; semi-gas fired; or straight gas fired. If it be direct fired, or semi-gas fired, the operation is too simple to require an extended explanation, it being merely a matter of gradually increasing the volume of fuel on the grate and the intensity of the combustion. With straight gas firing the operation is more tedious, since the combustion chamber must be heated independently before the gas is admitted. This has to be done by leaving out some of the retorts and erecting a temporary fireplace in their room, in which a wood or a coal fire is maintained until the interior of the combustion chamber is at least at dark red. Gas is then gradually admitted under constant observation of the flame, the producer having been put in operation in the meanwhile, but the preliminary fire is maintained until a strong regular draught through the combustion chamber is set up. When that condition has been established the temporary grate is removed, hot retorts from the tempering furnace are put in the vacant niches, and to replace those which may have cracked during the gradual heating, and the furnace is then ready for charging.

Mixture of Ores.—The mixture of ores for zinc smelting is far from being reduced to the scientific basis that obtains in lead and copper smelting and it is doubtful if it can be. The rules that we have are rather negative than positive. The object is to avoid the formation of fusible slags and the presence of compounds that will prevent driving off the zinc or that will excessively contaminate the spelter. With respect to the first point, it may be assumed that there will be present generally an excess of silica, which is derived not only from the ore but also from the ash of the coal; hence we must avoid having too much iron in the ore; and especially too much iron in the presence of lime and sulphur; manganese plays the part of iron, but is even worse. With respect to the second point, we should, for example, avoid ores containing magnesium sulphate, which is easily formed in the roasting of dolomitic ores, since in the retort this compound gives up some of its sulphur to zinc. Calcium sulphide does not appear to be injurious and calcium oxide may be positively advantageous in decomposing zinc sulphide, if its advantage in that respect be not offset by the presence of too much iron with which it will form a corrosive slag. Too high a percentage of lead in the ore is undesirable because it will excessively contaminate the spelter, even if it does not contribute to the

destruction of the retorts; or else the lead will have to be prevented from distilling over by more moderate driving of the furnace, at sacrifice of some of the available zinc.

Admitting the above conditions to be correct, we are still unable to reduce them to numerical factors, perhaps because they have not yet been sufficiently the subject of experimental and analytical investigation and perhaps because they are so complicated as to be incapable of a reduction to empirical rules.

Desirability of Uniformity in the Charge.—In practice the mixture of ores is based chiefly on the results of experience. In Europe, where the smelters receive ores of very diverse character, it is sought to incorporate a variety of different products of which the injurious constituents of each will neutralize themselves. A mixture having been made which gives satisfactory results with the retorts that are available, it is aimed to conform subsequently to that mixture so far as possible, in order to maintain the uniformity of the distillation process that the furnacemen are familiar with. If in exceptional cases there are ores with a particularly bad gangue, which have to be smelted, they are used in the mixture only in very small proportion, according to the indications of experience.

In the United States the conditions are radically different, the smelters being of two classes: (*a*) Those who mine and smelt their own ore; and (*b*) those who buy ore in the open market. The former have ore of uniform character to which they are able to adapt their smelting process. The latter buy ore from one or two districts, for the more part from one district, and are able to obtain products which are not only of rather even character, but also are remarkably free from deleterious impurities. The Kansas and Missouri smelters command a supply of blende averaging nearly 60% Zn, 1 to 1·5% Fe, and 0·5 to 1% Pb.; and of hemimorphite assaying 40 to 45% Zn, 0·5 to 1% Fe, and only a trace of lead. The smelters of Indiana and Illinois command the same ore, and also a small supply of blende and smithsonite from Iowa and Wisconsin, whence the blende contains somewhat more iron than the Joplin ore. The Joplin district affords a good deal of comparatively inferior blende, which the smelters are able to buy on advantageous terms, but there is no great difficulty in securing a mixture that will assay only 2 to 2·5% Fe, and save for the small percentage of lead the gangue is chiefly quartz.[1]

[1] Since this paragraph was written, the conditions have changed materially through the appearance in the market of large supplies of ore from the Rocky Mountain States and British Columbia, which some of the smelters have been using more or less extensively in admixture with the Joplin ore. The ore from the Far West is of various classes, but generally is lower in zinc and higher in lead and iron than the Joplin ore.

Mixture of Zinc Silicate and Roasted Blende.—The Western smelters frequently make a mixture of roasted blende and raw silicate on the theory that a better yield of zinc is obtained from the two together than from either alone. It is probably true that the mixture will give a higher yield than would be obtainable from the silicate alone, inasmuch as the charge is richer in zinc, but that the admixture of silicate improves the yield from the blende cannot be regarded as proved.[1] There are works which use roasted blende alone and recover as high a percentage of metal as any works which smelt a mixture, or higher. Such works as smelt raw silicate alone fail to make a high recovery of metal, a result that would be naturally expected. At certain works, which made a practice of smelting a mixture, a charge of 70% roasted blende and 30% raw silicate is frequently employed; the proportions are sometimes 90% and 10%, 80% and 20% and rarely 50% and 50%.

Following is an example of a charge smelted in Kansas: The mixture consisted of 70% roasted blende and 30% raw calamine and averaged 58% Zn, 1·25% Fe and 1·04% S. In distilling it, together with the blue powder, skimmings, etc., of a previous day (amounting to about 40% of the weight of the new ore) there was used 4550 lb. of reduction material per 10,000 lb. of ore. The reduction material assayed 8·41% water, 21·64% volatile matter, 56·45% fixed carbon and 13·5% ash. The residues drawn from the retorts amounted in weight to 48% of the weight of the ore and assayed 7·12% Zn.

The proportion of silicate ore now used by the Kansas smelters is less than formerly and the number of smelters who do not employ it at all is greater. This is due partly to the relatively greater increase in the supply of blende[2] and partly to commercial conditions.

Grade and Character of Ore Smelted in Belgium.—The average grade of the ore distilled by eleven Belgian works in 1898, reported directly by the various smelters, was given by Ad. Firket in *Annales des Mines de Belgique*, VI, i, 34, as follows, the figures of one company operating three works

[1] According to Kerl, Grundriss der Metallhüttenkunde, p. 448, the use of 15% to 20% of calamine in admixture with the roasted blende facilitates the distillation.

[2] An idea of the relative proportion of blende and calamine available to Kansas smelters may be obtained from the following statistics: In 1901 the Kansas portion of the Joplin district produced 33,978 tons of ore (Mineral Resources of Kansas, 1900 and 1901). The State of Missouri produced 207,234 tons of blende and 16,839 tons of "silicate" (Fifteenth Annual Report of the State Lead and Zinc Mine Inspector). The total production of zinc ore in Kansas and Missouri was therefore 258,051 tons, practically the whole of which was from the Joplin district. The proportion of silicate in the total was about 6·5%. However, a good deal of the blende was used by Illinois smelters, who do not take any silicate from the district. Estimating their purchases at 75,000 tons, the proportion of silicate would still be less than 10% of the remainder.

Works.	A	B	C	D	E	F	G	H	I
Elements.	%	%	%	%	%	%	%	%	%
Zn	47·00	46·60	43·00	41·60	45·00	53.00	49·36	50·00	46·50
Pb	3·80	6·60	1·80	5·94	8·00	6.50	5·26	5·50	7·90
Fe[a]	10·34	5·50	8·40	6·04	19·00	?	9·37	8·50	10·40
Ag	0·006	0·006	0·065	0·01	0·01	?	0·005	?	0·044
Cd	0·005	0·06	0·10	nil	?	?	?	?	0·215
Cu	tr.	nil	0·15	?	0·20	?	0·025	?	nil
S	1·08	1·80	1·50	1·76	1·20	?	1·28	?	2·95
As	tr.	tr.	0·05	nil	?	?	?	?	0·016
Sb	tr.	nil	0·03	nil	tr.	?	0·05	?	0·035
Ca	4·00	2·80	3·50	?	6·00	?	2·12	4·00	5·25
Mg	0·60	0·80	1·20	?	1·00	?	1·17		
SiO_2[b]	10·00	13·80	22·00	?	10·00	?	11·03	6·00	13·85

a Includes manganese. *b* Includes insoluble matter other than silica.

The especially noteworthy features of the above analyses are the comparatively low grade in zinc of the ore smelted and the high tenors in iron and lead from the American standpoint.

Limit of Iron in the Charge.—In general, ores with a high percentage of iron are not desired by zinc smelters. Nevertheless certain blendes containing after roasting 20% ferric oxide, about 1% manganese oxide, and 6% silica are treated by the Société Anonyme de la Vieille Montagne without difficulty, and under normal conditions. Those particular blendes can be well desulphurized, however, and after roasting are easily reducible. Low grade calamines assaying 40% ferric oxide have been smelted, exceptionally, with good results in so far as the consumption of retorts is concerned. On the other hand some calamines with a high percentage of ferric oxide, but of a very refractory character, have given very bad results. When the ore is very ferruginous, and at the same time contains 2 to 3% S, it may be expected that the smelting will give bad results, and a large consumption of retorts.[1] In brief, this question is dependent especially upon the character of the ore and the temperature which is required for its reduction. However, I think it may be said conservatively that a European smelter would rarely care to have more than 10% iron in his charge. In New Jersey the ore which is smelted nowadays contains 2 to 3% Fe and 4 to 5% Mn. Formerly it contained 2 to 4% Fe and 6 to 7% Mn, the limit of Fe+Mn being set at 11%. In Kansas and Missouri the percentage of iron that is tolerated is much lower. With the exception of the Cherokee-Lanyon Spelter Co., which is using Sadtler retorts, there are probably but few smelters who carry more than 2·5% Fe in their charges, and important reductions in the value of the ore are made when the tenor in iron exceeds

[1] De Sinçay, The Mineral Industry, VIII, 657.

1 or 2%. The successful distillation of a charge high in iron implies among other things the use of a superior quality of retorts. With inferior retorts only a moderate percentage of iron will lead to disastrous results.

It is customary in the West to deduct from the value of a ton of ore 50c.@$1 per unit for iron in excess of 1 or 2% to cover the extra cost of smelting and lower extraction of metal due to the presence of iron, or what is the same thing to offset the relatively higher cost of superior ore that must be bought to produce the desired smelting mixture. A deduction of $1 per unit is probably too high and unfair to the miner, 50c. more nearly representing the actual difference to the smelter. Theoretically the deduction for iron ought to vary with the market price of spelter, but the small differences due to its fluctuations are usually ignored. They can be covered, however, by deducting a certain percentage from the weight of the ore; for example, if a price were made for ore containing not to exceed 1% Fe, a lot assaying 2% Fe might be settled for on the basis of 97·5% of its actual weight and a lot with 3% Fe at 95%, and so on.

Limit of Lead and Its Effect.—With respect to lead in the ore mixture, the Société Anonyme de la Vieille Montagne prepares its charges so that they never contain more than 3 to 5% of that metal, since its metallurgists consider that a higher percentage rapidly destroys the retorts;[1] as to the degree of such effect of lead, opinions differ somewhat (vide p. 212). The charges for furnaces which are to be driven hard should not contain more than 3% Pb; but in those which are to be smelted at a lower temperature, the tenor in lead may rise to 6%, and even to 8% or more.[2] There are several works in Belgium which make a practice of treating ores high in lead; they generally run their furnaces at comparatively low temperature, sacrificing something in the recovery of zinc, but gaining in lead, the major part of which remains with the residues in the retorts. The residues are subsequently enriched in lead by jigging. With ore assaying 6% Pb it may be estimated that 10 or 12% of the quantity present will be distilled over and condensed. This is to say, if 1000 kg. of ore assaying 45% Zn and 6% Pb were distilled there might be produced approximately 400 kg. of spelter containing 7·2 kg., or 1·8%, of lead. The lead tenor of the crude spelter produced in Belgium sometimes attains 2% and more; and in Upper Silesia it is frequently 3%; but when the percentage of lead in the

[1] De Sinçay, loc. cit.

[2] This appears to represent the general practice in Belgium. There is, however, one works in Europe where ore containing 14% Pb is smelted regularly. Its retort residues assay 18·5% Pb, and its spelter only about 1% Pb. Refer also to the experience at the Emu works, in Wales, described in Chapter XVII, where ore assaying 24% Pb was distilled, the spelter averaging only about 0·5% Pb. This would imply a comparatively low temperature of distillation, which inference is supported by the comparatively high zinc tenor of the residues.

ore passes a certain limit the proportion which distils over appears to be less. Conversely, when the percentage of lead in the ore is very small and the percentage of zinc is high the proportion of lead which goes over into the spelter may be large. For example, in distilling Missouri blende containing 1% of lead we are apt to obtain a spelter assaying 0·5 to 1% Pb, indicating that from 25 to 50% of the lead contents of the original ore have been distilled and condensed with the zinc. On the other hand a yield of 150 kg. of spelter assaying 3% Pb from 1000 kg. of ore containing 200 kg. of zinc, corresponding approximately to the results obtained in Silesia, shows that 4·5 kg. of lead, or 0·45 unit of lead in the ore, has passed from the ore into the spelter. As previously remarked, the distillation of lead depends a good deal upon the temperature; thus it is possible to obtain a metal of lower lead content than the average by keeping separate the first drawing, which is metal distilled at the lowest temperature.

In Kansas and Missouri, where ores with more than 2% or 3% Pb are rarely offered, it is customary to deduct 50c.@$1 per unit in excess of 1%. It is doubtful, however, if the actual difference to the smelter because of the presence of lead in such ores would be more than 25@37·5c. per unit and under certain circumstances might be less than that, and even nothing at all. In so far as contamination of the spelter is concerned, a tenor of 1% lead in the ore may be practically as bad as 3% and in either case the proportion of lead that will be contained by the spelter will not unfit the latter for many of the purposes for which required, and moreover its lead contents are paid for at the price of spelter. In the case of spelter very high in lead the maximum detraction in value, as compared with that of a spelter containing originally about 1% Pb, is the amount of the cost of refining, which is a comparatively simple process.

Reduction Material.—There is a considerable variety of carbonaceous substances which may be employed as reduction material. Among these are lean bituminous coal, anthracite, oven coke, retort coke, and cinder (partially degasified coal, e.g., locomotive head-end cinder). The part which is played by the reducing agent and the reactions which take place with the various kinds have been explained in a previous chapter. Experience has shown that it is important that the reducing agent should be low in volatile matter, but yet not entirely destitute of it. These requirements are met very well by anthracite coal, and where the latter is obtainable cheaply there is perhaps no better reducing material, but such localities are few. A specially prepared cinder, or one produced as a by-product, also may meet the requirements well. So too may certain natural lean coals. Where none of the above materials are obtainable a mixture of coal and coke may be

made. Coke alone does not give the desired result; it appears that the presence of a certain quantity of hydrocarbons is necessary. These hydrocarbons are of course distilled off before the reduction of the zinc begins.[1]

In New Jersey, Pennsylvania, Indiana and Northern Illinois the smelters use anthracite culm, of which the original supply is in close proximity to the smelteries of the Eastern States, while the Indiana and Illinois smelters are able to buy cheaply the screenings of coal handlers in Cleveland and Chicago, whither anthracite is shipped in large quantity for domestic purposes. In Kansas and Missouri the smelters employ a mixture of coke made from the local coal and a lean coal from the levels of the coal measures that are near the surface; two parts lean coal to one part of coke, or equal parts of each are common proportions.

In Belgium and Rhenish Prussia and Westphalia certain lean coals are generally employed. In Upper Silesia the reduction material for mixture with the ore has to be prepared artificially, since all the coal of that region, even the leaner sorts, is rather rich in gas. At the beginning of the zinc industry there, charcoal was employed as reduction material, but the smelters soon substituted for that a mixture of coke and cinder, the latter being the unburned material dropping through the grates of direct fired furnaces. As the demand for that material increased, it is not unlikely that many furnaces were run for a good production of cinder rather than for maximum economy in combustion of the coal. With the general introduction of gas firing the production of cinder became insufficient to meet the requirements of the zinc smelters and the latter were obliged to prepare it artificially, which is the general practice at the present time. This is effected in furnaces heated sufficiently to drive off most of the volatile matter of the coal, without converting the latter into coke, experiments on the use of coke alone as reduction material having been unsatisfactory. These cinder-producing generators are so arranged that the gas distilled from them can be utilized for heating boilers, etc., and in some plants their ammonia contents are recovered. The degasified coal is drawn out between the grate bars. Other

[1] The metallurgists connected with the works at Overpelt, Belgium, consider that the presence of hydrocarbons in the reduction material plays a very important part in the distillation of refractory ores, the hydrocarbons being decomposed and depositing finely divided carbon around every particle of ore, thus preventing the fusible impurities from slagging and promoting the reduction of the zinc oxide. With this in view, Mr. Wilhelm Schulte, of Overpelt, has patented the process (United States patent, No. 718,222, of Jan. 13, 1903) of impregnating the ore with 3% to 5% of tar previous to mixing with the coal, whereby it is claimed that the quantity of coal may be greatly reduced, 25% in connection with the tar being as effective as 50% without it. This greatly increases the capacity of the furnace, and consequently reduces the cost of smelting by 20% to 30%, it is claimed. The use of tar for briquetting the charge is by no means new, having been proposed by Binon and Grandfils, vide p. 510, who had much the same objects in view.

works employ externally heated ovens (Backöfen) in which the coal is baked, caking somewhat according to its quality. Only Grieskohlen (rice) or Erbskohlen (pea) are used for making cinder, the yield of which ranges from 63 to 72% of the weight of the raw coal, according to the character of the latter.

Requisites for the Reduction Material and Its Functions.—Irrespective of the character of the reducing material the following rules concerning it may be laid down: It should be low in sulphur; low in moisture; and low in ash. The reasons for these requirements have been explained fully in previous sections of this treatise.

The reducing material performs not only the function of reducing agent, but also that of a sponge to absorb the slag formed by the gangue of the ore. Because of the difference in the reducibility of various ores and their different slag forming tendencies the proportion of reducing material to ore must be varied according to the ore and the correct proportion can be determined only by experience. In any case the quantity of coal required will be greatly in excess of what is demanded by the theory. Theoretically, 1 kg. of zinc oxide should be reduced by 0·148 kg. of carbon; practically, about three times that quantity will be used at least.[1] The presence of so great an excess of carbon is necessitated by the requirement of an intimate mixture of ore and coal and the insurance that there will be an unburned surplus which will absorb the slag and prevent it from draining through the charge to the walls of the retort, possibly entraining particles of unreduced zinc oxide on the way.

These actions hang together to a large extent. Every particle of zinc oxide must be exposed directly to carbon, or carbon monoxide, until it is reduced. There must be therefore a surplus of carbon so that every particle of zinc oxide will certainly be exposed to its action and the longer the time it must be exposed the greater must be the excess of carbon to insure that there will be active carbon present; and the greater the quantity of slag that will be formed the greater must be the quantity of carbon because the efficiency of the latter will be reduced by the slag which will envelop it. The fact that the carbon becomes thus coated with slag makes comparatively valueless the excess that may be recovered from the retort residues by washing, while the function that the carbon performs as a sponge is illus-

[1] Assuming that a roasted ore assaying 70% Zn be presented for distillation, 100 kg. will contain approximately 87 kg. of ZnO for the reduction of which there will be required theoretically 87×0·148=12·876 kg. of pure carbon=14·3 kg. of coke or cinder if the latter contained 90% carbon. Practically, at least 40 kg. of coal would be used, which is about three times the theoretical quantity. With ore of lower grade the proportion is higher.

trated when an insufficient quantity is used and there is a large accumulation of slag on the bottoms of the retorts, corroding them and leading to increased loss of metal.

Quantity of Reduction Material Required Practically.—In practice the quantity of reduction material that is required is found to be from 33⅓ to 66⅔% of the weight of the ore. It is rarely that either of those extremes are employed and in general it may be considered that 50% is the maximum and 40% the minimum. The modern tendency in zinc smelting is strongly, however, toward the use of an increased percentage of reduction material. It is nevertheless obviously desirable to use the smallest quantity that is consistent with a good extraction of metal, not only because of the direct cost of the reduction material, but also because it occupies room in the retorts that would otherwise be available for ore. In the latter connection it may be pointed out that the crushed coal will weigh in the neighborhood of 50 lb. per cubic foot, while the blende assaying 56 to 58% Zn of the Joplin district will weigh about 125 lb. after roasting. When 40% of reduction material is used its volume and that of the ore are therefore precisely equal.

At a certain works in Belgium, where ore averaging 47% Zn, 3·8% Pb, 10·34% Fe+Mn and 10% SiO_2 was distilled in 1898, the average percentage of reduction material employed was 41; it contained an average of 8·5% volatile matter, 9% ash, and 82·5% fixed carbon. At another works where the average charge assayed 46·60% Zn, 6·60% Pb, 5·50% Fe+Mn, and 13·80% SiO_2 there was used 46% of reduction material, which averaged 18% ash, 8% volatile matter and 74% fixed carbon.[1]

Mixing the Charge.—The ore which is to be charged into the retorts of a Belgian and Rhenish furnace must be in a state of fine subdivision, not to exceed 2 mm. size for the roasted blende and approximately 3 to 4 mm. size for calamine. The blende has already been reduced to the proper size as a preliminary to roasting. Inasmuch as there is no danger of sintering an ordinary blende ore in the roasting furnace, the product of the latter is in a condition to go directly to the distillation furnace as soon as it is cool enough to handle. The calamine has to be crushed to the requisite size, directly as it comes from the mine if it is to be used raw, or as it comes from the kiln if it be calcined. The Western smelters of the United States universally now use the hemimorphite of Missouri in a raw condition. In Upper Silesia, where smithsonite still forms nearly half of the charge (formerly it constituted the whole), it is crushed only to hazel nut size, the comparatively great height of the charge in the Silesian muffle requiring

[1] Ad. Firket, Annales des Mines de Belgique, VI, 1, 45.

that there be ample interstices for the escape of the vapor. For the same reason the cinder, coke or coal to be mixed with the ore is broken coarser in Upper Silesia than elsewhere. In Kansas and Missouri the coal and coke are reduced to approximately half inch size as a maximum. In Belgium it is the common practice to reduce it to a smaller and more uniform size.

The crushing of the coal and coke is a very simple matter, the softness of the material necessitating nothing but a machine of comparatively light construction. In Kansas and Missouri four-roll crushers are commonly employed, the upper pair of rolls being usually corrugated and the lower pair smooth. Sometimes both pairs are corrugated. Such a machine, built

FIG. 354. FOUR-ROLL COAL AND COKE CRUSHER.

by Friedrich Krupp, is illustrated in the accompanying engraving. Of course there are numerous other machines which are adaptable to this purpose, such as the Stedman (Carr) disintegrator and some of the coal crushers which are made by the Jeffrey Manufacturing Co., of Columbus, Ohio, and other concerns in the same line of business.

The constituents of the charge, comprising for example (*a*) roasted blende, (*b*) calamine, (*c*) lean coal, and (*d*) coke, are generally in small American works deposited in given quantities on the floor in front of the furnace, into the retorts of which they are to be put the next day. They are usually distributed during the afternoon and their mixture forms part

of the work of the maneuver of the next day. This was formerly the general practice in Europe also and may be still to some extent. In the large works of Belgium and the United States, however, the mixture of the charge is now commonly effected by mechanical means. At the Vieille Montagne works the ore and coal are ground together in Vapart mills; at Lasalle, Ill., they are mixed in a revolving hexagonal barrel; at Iola, Kan., they are shoveled together into a screw conveyor, which effects the mixture in the same manner as do the pug-mills that are sometimes employed in Europe for the same purpose. In each case the mixture is delivered in cars to the furnaces and the maneuver is relieved of that much labor.

Screw Mixer.—The screw conveyors, which are generally employed at Iola, are of the common design. They are set in the floor of the mixing house, the upper edges of the trough flush with the floor, and deliver the mixed charge into the boot of a belt elevator, by which it is transferred into the cars which go to the furnaces. It is worth remarking that when a screw conveyor is arranged in the manner described above the trough should be covered by a grating with openings sufficiently small to prevent the entrance of a man's foot. Some serious accidents have happened, in connection with open troughs, by men working near them slipping and becoming entangled by the screw.

Vapart Mill.—The Vapart mill is a centrifugal grinder, which is shown in the accompanying engraving. The material to be crushed, charged in through the top, arrives on the upper table, which is fixed to a rapidly revolving vertical shaft. It is thrown by centrifugal force, and the angle bars bolted to the table, against the corrugated lining of the cylinder. Between the corrugated lining bands there are chutes, which discharge the falling particles upon the next lower table. The finished product is discharged from the bottom of the cylinder in the same manner. Access to the interior of the cylinder is provided as shown in the accompanying engraving. This machine is manufactured by C. Mehler, of Aachen, in four sizes: No. 1, with cylinder 1·75 m. in diameter, requiring 15 to 18 h. p.; No. 2, 1·3 m.—8 to 12 h. p.; No. 3, 1·05 m.—5 to 8 h. p.; and No. 4, 0·8 m.—3 to 4 h. p. The list prices are respectively 6400, 4000, 2400 and 1600 marks (one mark=23·8c.). Vapart mills are used for various purposes at the Vieille Montagne smelteries at Angleur, Valentin-Cocq, Flône, Viviez and Berge-Borbeck; also at one of the works at Stolberg.

Moistening the Mixture.—However the ore and coal be mixed, it is customary to moisten the mixture with water before charging it into the retorts. This is done to make it easier to throw in the charge with the small scoops that are necessarily employed and to avoid raising dust during the charg-

ing. The charge is moistened to about the same extent as in preparing a brasque, i.e., so that a ball compressed in the hand will hold its shape, but will not adhere to the hand.

Preparation of the Charge in Cartridge Form.—Binon and Grandfils proposed to mix the charge for distillation with the necessary proportion of reduction material and a small percentage of tar, then molding it into blocks, whereby they claimed that 50% more ore could be introduced into the retort than by the ordinary method. It was further claimed that in this

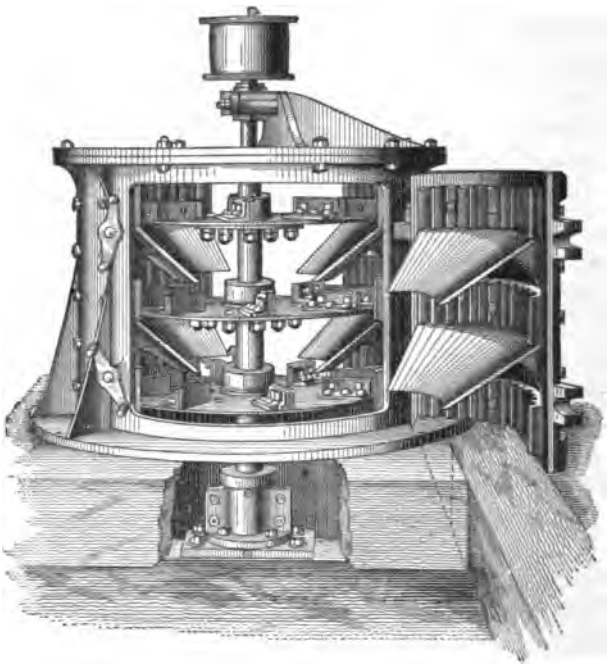


FIG. 355. VAPART MILL.

way the distillation could be effected at a comparatively low temperature, that a smaller percentage of reduction material would suffice, that breakage of retorts would be diminished, and that there would be a reduced consumption of heating coal per ton of ore, inasmuch as there would be no increase in the quantity required per furnace, while the furnace charge of ore would be considerably greater.[1] The fundamental idea of Binon and Grandfils was that the reduction of zinc oxide is effected by carbon, not

[1] Berg- u. Hüttenm. Ztg., 1880, 8; 1881, 27; 1882, 531; 1883, 198 and 211.

carbon monoxide, and it was therefore desirable to effect as intimate a contact as possible. Their method was tried practically at Bleyberg, Belgium, in 1882, with successful results, it was said, but it did not survive.

Recently a process which is substantially the same as that of Binon and Grandfils has been patented by H. F. Kirkpatrick-Picard, of London, who in connection with H. L. Sulman has been exploiting it as a means of smelting the mixed sulphide ores of Broken Hill, N. S. W.[1] An account of some results obtained in Wales with this method of smelting may be found in Chapter XVII.

The ideas of Wilhelm Schulte as to the advantages to be derived from the use of tar, or similar material, in connection with the ordinary reducing agent have already been referred to in the foot-note to p. 505.

Leo Lynen has proposed to make up the charge in pasteboard cartridges, which can easily be shoved into the retort, claiming thereby that not only is the labor of charging reduced, but also that the carbonized pasteboard will form a protective lining for the retort. I am unaware that this method has ever been tried. It would be rather expensive and probably would be difficult to manage.

LABOR IN DISTILLATION.—The amount of labor required for the management of a distillation furnace is governed chiefly by the capacity of a certain number of men to perform the maneuver of a certain number of retorts in a period of hours which will leave ample time for the distillation of the ore without butchering the retorts. There is a direct relation between the time consumed in the maneuver and the extraction of metal from the ore, so that an economy in labor may be offset by an increased loss of metal and vice-versa. Before attempting to draw any conclusions, however, it is best to cite certain general results of practice.

Crew for Direct Fired Belgian Furnaces.—The crew of the $2\times7\times16=$ 224 retort Belgian furnace used in the western districts of the United States is composed of two brigadiers (or chargers as they are more commonly called in this country), two smelters (also called stokers and "long shifts") and four helpers (short shifts). The crew is divided into two sections, one for each side of the furnace. The duty of the crew is to mix the ore and coal, perform the maneuver, conduct the distillation, draw the metal and pile the slabs ready for removal and discharge the residues into the collecting chutes. It has nothing to do with the fetching of the ore and coal to the furnace, or the removal of residues and metal from it.

It is an old established and commonly followed custom for the brigadiers

[1] British patents, 2151, 15,166 and 22,570 of 1900; United States patent, 665,744, Jan. 8, 1901.

and smelters to work a double shift, or from 6 a. m. to 6 a. m., 24 hours, with a day off in between. The helpers work short shifts each day. The idea in having the brigadiers and smelters on duty for 24 hours continuously is that the working of a charge occupies that time, and in having it performed by one set of men it is possible to fix the responsibility for negligence and maintain the excellence of the work by awarding prizes or premiums for yield of metal in excess of a certain standard; or what amounts to the same thing, the men may be paid a certain price per pound of metal obtained. Such an organization of labor is desirable, inasmuch as the personal factor enters largely into the success of the distillation process and a careless furnace crew may offset the advantages of the best metallurgical improvements that are possible.

In the operation of the 224 retort Belgian, direct fired furnace, there are required therefore:

2 brigadiers, 2 shifts	4
2 smelters, 2 shifts	4
4 helpers, 1 shift	4
Total number of shifts	12

The number of retorts being 224 and the total number of shifts 12, and $224 \div 12 = 18{\cdot}667$, one man-shift attends to 18·667 retorts per 24 hours; but during the maneuver there are eight men at work, wherefore since $224 \div 8 = 28$, one man attends to 28 retorts. In Kansas and Missouri the wages paid the furnace men are approximately as follows: Brigadiers, $4·70 per double shift; smelters, $3·80 per double shift, and helpers, $1·40 per single shift. The daily cost is therefore:

2 brigadiers @ $4·70	$9·40
2 smelters @ $3·80	7·60
4 helpers @ $1·40	5·60
Total	$22·60

The 224 retort furnace will smelt from 4·8 to 5 tons of roasted Joplin ore assaying 57 to 60% zinc before roasting (66 to 70% after roasting), together with all the blue powder, oxidized skimmings and sweepings from the previous charge. The labor cost of distillation is consequently $4·52@ $4·71 per ton of roasted ore.

The furnace crew specified above is considered in the West to be the smallest that can handle a 224-retort furnace. With the $2 \times 7 \times 14 = 196$ retort furnaces there will be the same number of men, but they may be paid

only $4·50 (brigadiers), $3·60 (smelters), and $1·30 (helpers). The 2×8×16=256 furnace will have two extra helpers, which will give 18·3 retorts per man-shift and 25·6 retorts per man during the maneuver.

Crews for Long, Gas Fired Furnaces.—With the long furnaces employed at the works of Matthiessen & Hegeler at Lasalle, Ill., the charging is done by tiers of retorts from one end of the furnace to the other. The furnaces being gas fired the labor of stoking is concentrated at the producers and after the maneuver there is no labor required on the furnace except for regulation of the distillation and drawing of the metal. The labor is so organized that there are no double shifts. The crew of a 864 retort furnace consists of the following:

Day: 1 stoker; 1 stoker's helper; 4 metal drawers (two per side); 8 retort cleaners; 4 chargers (two per side); 2 men setting condensers (one per side); two men mixing blue powder, etc. (one per side); 4 boys cleaning condensers (two per side); total, 26.

Night: 1 stoker; 1 stoker's helper; 4 metal drawers (two per side); total, 6.

The furnace is therefore operated by 32 man-shifts, or one man-shift per 27 retorts, and one man per 33·23 retorts during the maneuver. There is a natural saving in labor because of the concentration of the stoking at the producers and because the ore is mixed previous to its dispatch to the distillation furnaces, but these do not account for the great number of retorts managed per man during the maneuver, which is due rather to the more systematic manner in which the work is performed, to the various labor saving devices that have been introduced, and perhaps also to a slight prolongation of the time of the maneuver.

The method of operating the long, natural gas fired furnaces at Iola, Kan., is similar to that followed at Lasalle, Ill., the furnaces being charged in tiers of retorts from end to end and the men working single shifts. The crew of a 620 retort furnace and the wages paid at Iola are as follows:

Day:	1 fireman	$2·75 to $3·00	
	4 chargers, $2·30 @ $2·35	9·20 "	9·40
	6 men in front of the car, $1·40 @ $1·50	8·40 "	9·00
	4 men behind the car, $1·60	6·40 "	6·40
	2 condenser cleaners, $1·70	3·40 "	3·40
	17	$30·15 "	$31·20
Night:	1 fireman	2·75 "	3·00
	2 metal drawers, $2·35	4·70 "	4·70
	1 stuffer and metal piler	1·50 "	1·50
	4	$8·95 "	$9·20
Total wages of day shift		30·15 "	31·20
" " night "		8·95 "	9·20
" " per 24 hours		$39·10 "	$40·40

The maneuver is begun at 5 a. m. or 6 a. m. and is finished at about 10 a. m. or 11 a. m., respectively. The men in front of the car correspond to the helpers on other furnaces; they remove the condensers, discharge the residues from the retorts, replace broken retorts, etc. The men behind the car replace the condensers after the retorts are charged, and otherwise attend to the distillation. Their work ends with the completion of the maneuver. The condenser cleaners scrape the crusts of oxide and blue powder out of the condensers and prepare them for further use; afterward they attend to the distillation, remaining on duty throughout the day.

The Iola furnace of 620 retorts is therefore operated by only 21 man-shifts per 24 hours and 17 men during the maneuver, which is one man-shift per 29·5 retorts and one man per 36·5 retorts during the maneuver. The large number of retorts per man-shift is of course accounted for in part by the use of natural gas, which practically eliminates the labor of stoking. In the early days at Iola the maneuver was apt to be greatly prolonged, of course with disastrous results, but better management has remedied that and in the latest practice the maneuver is completed in about 5 hours. It is customary in the summer, when the weather in Kansas is extremely hot, to begin it as early as 4 a. m.

The Iola furnaces are generally charged with 12·5 to 13 tons of ore. Assuming a labor cost of $40·40 per day, the average per ton of calcined ore is about $3·10@$3·25. The figures herein quoted are about representative of normal conditions, the wages for labor varying from time to time and the practice at various works differing in minor particulars.

Labor on Natural Gas Fired Furnaces in Indiana.—At Marion, Ind., the long, natural gas fired furnaces are worked in sections of 200 retorts, which are charged in rows. Each section has the following crew:

Day: 1 charger	$2·00
1 smelter (long shift)	1·75
4 helpers @ $1·35	5·40
6	$9·15
Night: 2 metal drawers @ $1·50	3·00
Total	$12·15

There are therefore eight man-shifts per 200 retorts, or 1:25, and six men handle 200 retorts or 1:33⅓ during the maneuver. The charge per 200 retorts is four tons, wherefore the labor cost in distillation is $3·04 per ton.

At Ingalls, Ind., the 2×4×53=424 retort furnace had the following crew:

Day:	2	brigadiers @ $1·90	$3·80
	2	smelters @ 1·75	3·50
	10	helpers @ 1·25	12·50
	14		$19·80
Night:	1	fireman	2·00
	4	metal drawers, @ $1·50	6·00
	5		$8·00

The number of retorts per man-shift was 22·3 and per man during the maneuver 30·3. The cost of labor per day was $27·80.

The 2×4×40=320 retort furnace had the same crew except that the number of helpers was only eight, making 18·82 retorts per man-shift and 26·67 per man during the maneuver, the daily cost for labor being $25·30. The furnace was charged lightly, 320 retorts receiving only 5·3 tons of ore, which made the cost come to $5·28 per ton of roasted ore.

Labor on Distillation Furnaces in Belgium.—At Angleur the massives comprising 400 retorts, gas fired by producers at each end of the massive, are manned during the maneuver by a crew consisting of one brigadier and two smelters per each quarter section of 100 retorts. There is also one stoker per end per shift. The maneuver begins about 5 a. m. and is finished about 10 a. m. During the distillation period one man per side remains to draw the metal, etc. There are therefore 14 men on the day shift, or one man per 28·5 retorts and four men on the night shift. The total number of shifts is 18, or one man-shift per 22·2 retorts. These do not include the trammers, who fetch coal and ore and wheel away ashes and spelter, nor boys who attend to the prolongs and aid the furnacemen in various minor ways. It will be observed that in these modern Belgian furnaces the efficiency of the labor is about equal to that attained in American practice, but the wages paid are much lower and the labor cost of distillation is very much in favor of the Belgian producers, and will no doubt continue to be so as long as zinc ore has to be distilled in small retorts. The advantage of the Belgians with respect to labor is offset, however, by the high cost of their coal supply.

The older direct fired furnaces of 140 retorts (70 per side) which are still used in Belgium to some extent are manned by two brigadiers and four smelters during the maneuver, and by two men during the distillation, i.e., one man per 23⅓ retorts during the maneuver and 17·5 retorts per man-shift for the entire 24 hours.

Conclusions as to Labor Requirements.—The foregoing data as to the labor employed in connection with the distillation process in Belgian furnaces are summarized as follows:

Locality	Method of firing	No. of retorts in furnace	No. of retorts per man-shift per 24 h.	No. of retorts per man during maneuver
Kansas	Direct	a224	19	28
Collinsville	"	a256	18	26
Lasalle	Producer gas.	a864	27	33
Iola	Natural gas.	a620	29	36
Indiana	" "	a200	25	33
"	" "	a424	22	30
Belgium	Producer gas.	b400	22	28
"	Direct	a140	18	23

a Double furnaces with two working faces, separated completely by middle longitudinal wall. *b* Massive of two double furnaces, each of which has two working faces but is not entirely divided by middle longitudinal wall.

It appears therefore that experience has shown that in the operation of Belgian furnaces there is required during the maneuver one man per 25 to 33 retorts. In that part of the process there is not necessarily a great difference between natural gas firing and direct coal firing, because in the case of the latter the fires require temporarily but little attention and the stoker helps the other smelters to discharge and charge the retorts. The maximum efficiency is attained with long furnaces, not more than five rows high, which are charged in tiers with the aid of cars and permit possibly of the best systemization of the labor. With respect to the entire period of 24 hours, however, the gas fired furnaces acquire a distinct advantage because the labor of stoking is dispensed with in the case of natural gas firing and is minimized in the case of producer gas firing, and the other labor being chiefly supervision of the distillation and drawing of the metal one man is able to attend to a relatively large number of retorts.

Labor on Rhenish Furnaces.—At Stolberg, Rhenish Prussia, a furnace with 240 retorts is manned by the following crew: Two stokers, each working a 12-hour shift; two metal drawers (lads of 16 to 18 years), each working a 12-hour shift; 14 furnace men working a short shift (during the maneuver), and one man on the day shift wheeling ore, coal and residues. Such a furnace with retorts 1·2 to 1·4 m. in length smelts about eight metric tons of roasted ore mixed with 40% of reduction coal per 24 hours, with a consumption of 8·5 to 9·2 tons of heating coal. The furnaces are gas fired and have heat recuperators (counter current system). Excluding the trammer, there is one man to 15 retorts during the maneuver and one man-shift per 13⅓ retorts during the 24 hours.

At Dortmund, Westphalia, a furnace with 240 retorts smelts 8000 kg. of ore, mixed with 30% of its weight of reduction material, per 24 hours, with

a crew consisting of two stokers, each working a 12-hour shift, four brigadiers, four smelters and eight helpers. Including the stoker, there is one man per 14 retorts during the maneuver. The latter begins at 5 a. m. and is finished at 11 a. m.

At Letmathe, Westphalia, direct fired furnaces with 75 retorts, 0·165 ×0·255 m., were in use in 1895. The charge of ore per furnace was 1950 kg., with which 38 to 40% of reduction coal was mixed. The furnace crew consisted of one man working 24 hours and two men working 10 hours,[1] which would correspond to one man per 25 retorts during the maneuver and one man-shift per 18¾ retorts during the whole 24 hours. It is to be noted, however, that in this case the retorts are considerably smaller and the charge per retort less than in the general Rhenish practice.

At Neumühl-Hamborn, near Oberhausen, the furnaces, which are gas fired, and have 252 retorts, 0·16×0·30×1·40 m., distil from 7600 to 8000 kg. of ore per day. The furnace crew comprises 12 men during the maneuver and two men during the distillation.[2] This is an average of 18 retorts per man-shift during the 24 hours and 21 per man during the maneuver.

Labor on Silesian Furnaces.—On the old, direct fired furnaces with 20 to 28 muffles, the crew consisted of one smelter, working a 24-hour shift, and two stokers, each working a 12-hour shift. The furnaces with 28 to 32 muffles have one smelter, one helper and two stokers. This corresponds to about one man per 10 muffles during the maneuver, and one man-shift per six muffles during the 24 hours. The furnaces with 56 to 64 muffles have two smelters, two helpers and two stokers, which would imply about 12 muffles per man during the maneuver and 7·5 muffles per man-shift for the whole 24 hours. In the large furnaces, which have two smelters, one of the latter acts as chief, or brigadier. In the case of the small furnaces, which have only one smelter, the crews of two furnaces help each other in heavy work, such as replacing defective muffles. The stokers work 12-hour shifts, beginning at 6 a. m. and 6 p. m. They have to attend to the fires and in the case of the Siemens furnaces have to reverse the valves every half hour; they also assist the smelters during the maneuver. It used to be the common practice in Upper Silesia, and may still be so, to pay the furnace men so much per centner of zinc produced, the price being divided unequally among the smelters, stokers and helpers in the order named. The ratio is such that smelters earn about 4 marks per shift; stokers, 3@3·25; and helpers 1·50@2. In the early days of the zinc industry in Upper

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 174.
[2] Schnabel, loc. cit., p. 177.

Silesia, smelters earned only 1 mark, stokers, 0·75 mark, and helpers 0·50 mark.

THE MANEUVER.—In the operation of the ordinary Belgian furnace the maneuver of discharging and recharging the retorts is begun immediately after the arrival of the new shift at 4, 5 or 6 a. m., as the case may be. The old shift has previously made the last drawing of metal and permitted the fires to die down, so that the furnace is only at red heat. The brigadier and charger begin immediately to mix the ore and reducing material on the floor, if that has not previously been done mechanically. In the meanwhile the helpers remove the condensers of the highest row. In order to approach them a long bench is moved in front of the furnace, or a platform hung by trolleys from the tie beams of the roof is moved up. A sheet iron shield, equal in length to that of the furnace and about three rows of retorts in height, suspended by counterweighted chains at each end, is

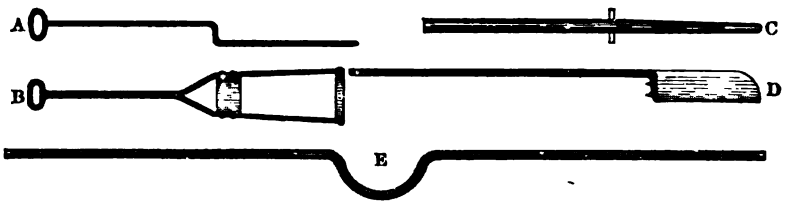


FIG. 356. TOOLS USED IN CONNECTION WITH BELGIAN TYPE OF FURNACE.

A. Bar for removing condensers. B. Holder for setting up condensers. C. Bar for setting up condensers. D. Charge scoop. E. Bar for carrying hot retorts from the tempering furnace.

pulled down to protect the man on the platform from the heat of the retorts below the row on which he is working. The man on the platform removes the condensers by means of a bent bar, like that which is shown at *A* in Fig. 356. He throws them one by one to his companion on the floor, who deftly catches them with a similar bar and deposits them conveniently on the floor. The first two or three inches of residue in the retort are then raked out and saved apart; this portion of the old charge, which is naturally higher in zinc than the average, is added to the charge of the next day following. The quantity obtained will depend somewhat upon the judgment of the smelters as to the extraction of zinc from the previous charge.

Cleaning the Retorts.—The rich residues having been removed, the retorts are ready for the discharge of the remainder. This is effected in various ways. The old system, which is still employed extensively, is simply to draw them out with rabbles. At the works of the Matthiessen & Hegeler

Zinc Co. at Lasalle, Ill., a platform car is employed for this purpose, as described in Chapter X. This arrangement is obviously advantageous only with long furnaces, which are worked in tiers and not in rows, or with series of short furnaces arranged in a straight line.

In Kansas and Missouri it has become the general practice to blow out the residues with steam. For this purpose the sheet iron shield in front of the furnace is provided with holes corresponding to the openings in the retorts. A 1-in. wrought iron pipe of suitable length, say about 8 ft., connected by a rubber hose with the main supply pipe, is put through the hole and thrust to the back of the retort. Steam is then turned on; or more commonly water is used, issuing from small holes in the end of the pipe. The water being suddenly converted into steam by the intense heat of the retort blows out the contents of the latter with an explosive effect. The hot ashes so blown out strike the shield hung in front of the furnace and drop into the open pockets below the floor. The loose ashes are blown out very cleanly and if there be no fused slag in the retort, the latter is ready for the next charge. However, there is generally more or less slag. The steam does not blow it out and it must subsequently be removed with the aid of a rabble in the usual manner. As to whether serious harm is done to the retorts by the use of steam or water in the manner described, it is an open question. With steam alone it may be indeed that the retorts do not suffer in any way, but with the use of water, the explosion which takes place is rather violent and it may be supposed to have a weakening effect upon the fragile walls of the retort. Some metallurgists are opposed to the use of water for this reason, calling attention to cracks in the retorts which are presumably due to it. The smelters like to use water rather than steam because the pipe is so much more easy to handle.

Charging by Rows.—The retorts of the uppermost row having been cleaned out, the working platform is moved again to the furnace and the brigadier and charger mount it to refill the empty retorts. One of the helpers throws the previously mixed charge from the floor to the platform, and the brigadier and charger throw it into the retorts by means of narrow scoops. Each shovelful is thrown in with considerable force, so that the charge is caused to fill the entire cylinder. Experienced smelters acquire great skill in thus filling these long cylinders of comparatively small diameter, each shovelful going unerringly to its destination, without any spilling of charge. With the ordinary 8×48 in. retort, from seven to nine shovelfuls are required per retort. Inasmuch as there are 16 retorts in a row, each man has to throw in from 56 to 72 shovelfuls per row; the work having to be done in the shortest possible time and the heat radiated from the

front of the furnace being high, the labor of charging is extremely arduous in character.

While the charging is going on the second helper scrapes the oxide and crust out of the condensers, which will be ready by the time that the charging is completed. The retorts having been filled, a small rod is thrust longitudinally through the charge at the top to make a hole for the escape of the first gas, the ore that has been spilled on the front shelf is scraped off and the helpers then replace the condensers, which are handled by means of a carrying bar like that which is shown at *C* in Fig. 356, or by means of the device shown as *B* on the same plate. One man sets the condensers in the retorts; another following him puts a piece of brick under each one to raise the end to the proper position. The joint between the condenser and the retort is then luted up and stamped tight with a red hot, crescent-shaped iron. Finally the open end of the condenser is closed with a ball of brasque. The second row of retorts is then discharged and recharged in the same manner, and so on with each successive row until the maneuver is completed, which will be about five hours after the beginning, depending among other things upon the number of broken retorts that have to be replaced. Only the seventh and sixth rows have to be charged from the platform; the charge for the fifth and lower rows can be thrown in directly from the floor of the furnace house. Consequently, with the modern furnaces, which are only five rows high, the platform is dispensed with.

Charging by Tiers.—In the long furnaces which are employed at Lasalle, Ill., and Iola, Kan., the retorts are handled in tiers, beginning at one end of the furnace and finishing at the other. The operations of the maneuver in this case are the same as when the retorts are handled in rows, but the method of handling them in tiers permits the various operations to be performed contemporaneously. Thus one gang of men will remove the condensers and rake out the residues; another gang following them will recharge the retorts; and a third gang following the chargers will replace the condensers. With this method, the charging is done invariably from a car which is moved on a track in front of the furnace. The duration of the maneuver depends upon the length of the furnace and number of men; with the $5\times60=300$ retort furnaces employed at Iola, Kan., and the crew with which they are ordinarily handled, it is generally completed inside of six hours. There is no doubt that the handling of a distillation furnace from end to end is advantageous with respect to economy in labor, but if the furnace be undermanned, through mistaken ideas as to economy, the advantages may easily be lost. End-to-end charging is not restricted, however, to furnaces of excessive length; it is not indeed advantageously adapted

to the ordinary 7×16 Belgian furnace, but with certain short furnaces the method can be employed quite as well as with the long Iola and Lasalle furnaces.

DISTILLATION.—After the completion of the maneuver the work of the helpers (short shifts) is ended for the day and the distillation of the charge is conducted by the smelters, or as they are variously called, the brigadier and charger, brigadier and stoker, charger and long shift, fireman and metal drawer, etc. Even before all the retorts have been charged, the development of gas has begun in the upper rows, or in the tiers adjacent to the near end of the furnace, according to the method of charging. Before this gas has acquired sufficient pressure to blow out the stoppers in the noses of the condensers, small holes are made in the latter to permit the escape of the gas, which by that time has sufficient tension to cause a lively whistle in passing through the outlet made for it. This gas is pure coal gas resulting from the distillation of the hydrocarbons in the coal, being analogous to ordinary illuminating gas without enrichment. It burns with a rather luminous flame, which becomes less bright as the percentage of carbon monoxide increases, gradually becoming purple and acquiring the other characteristics of CO gas. The maneuver having been completed, the temperature of the furnace, which has previously been maintained at a red heat, is rapidly raised to white heat. After a short time the flames issuing from the condensers become tinged bluish green from the presence of burning zinc, indicating that the distillation has begun. As the distillation increases in activity the flame becomes more luminous and brilliantly bluish green, and so continues until toward the end of the period when it gradually disappears and is succeeded by the purple carbon monoxide flame. The burning zinc produces a whitish smoke. If cadmium is present it will be detected by the appearance of a brownish smoke, particularly at the beginning of the distillation, when the cadmium chiefly comes over. If prolongs are employed, they are put on the condensers when the zinc flame first appears.

During the early part of the distillation period the zinc which comes over is condensed largely as blue powder and oxide, because of the relative coolness of the condensers, the poverty of the gas and the presence of some carbon dioxide in it. The zinc will not condense readily as liquid until the percentage of zinc vapor has increased and the diluting gas is almost pure carbon monoxide. Even then there will be a considerable formation of blue powder if the temperature of the condensers be too low; on the other hand, if it be too high an increased percentage of zinc will escape without condensation.● It is the duty of the brigadier to regulate this. If the wind

be fresh, the doors on the windward side of the furnace house are closed to prevent an undue cooling effect on that side of the furnace. At Lasalle, Ill., the furnace houses, which are there very large, are provided with suspended sheet iron shields, which can be pulled down so as to enclose the furnaces on all sides, the shields in front of the façades being about 6 ft. distant from the latter.

Besides regulating the temperature of the condensers, the brigadier sees that they are kept clear of oxides and crusts, which might accumulate so as to obstruct the free egress of the gas, and watches closely for broken retorts. The existence of the latter is indicated by a dying out of the flame at the nose of the condenser, the chimney draught drawing the gas through the crack into the combustion chamber, wherefore the pressure inside of the retort is diminished. This is indicated also by the appearance of a white smoke issuing from the chimneys during intervals between stoking, when its presence is not concealed by the great volume of black coal smoke. As soon as a defective retort is located, the metal in its condenser is drawn off, the condenser is removed, and the contents of the retort are pulled out.

The condensed metal is tapped three times during the twenty-four hours, usually at 2 p. m., 10 p. m. and 5 p. m. It is drawn from the condensers into a kettle, which is swung under the condenser noses so that the liquid metal accumulated in the condenser can be readily scraped out into the kettle. As soon as a kettleful of metal has been obtained, a handful of coke dust is thrown in, to prevent undue oxidation of metal, and the oxide and blue powder are then skimmed off and reserved for redistillation with the next day's charge. The clean metal is then poured into the molds, which have conventionally the form of plates, or slabs. The dimensions and weight of these slabs vary somewhat at different works; a slab weighing 50 lb. is an ordinary size. This requires a mold $9\times18\times1\frac{1}{4}$ in., which will hold 52 lb. even full; one of $9\times18\times1\frac{3}{8}$ in. will hold 57 lb. even full.[1] Some smelters make slabs weighing only 45 lb., and others make them weighing as much as 65 lb. At Iola it is the general practice to cast the ordinary spelter, known as "prime Western," in 65 lb. slabs; the first drawing of metal, which is the purest because of having been distilled at the lowest temperature, is frequently cast in 50 lb. slabs and sold as "brass special," "extra select," or under some other particular trade name. In Belgium a common weight is 18 to 20 kg.

Practice in Belgium.—The method of performing the maneuver and conducting the distillation with the various types of furnaces employed in Belgium is essentially the same as in the United States. The furnace crew,

[1] A cubic inch of cast zinc weighs 0·2564 lbs.

or brigade, consists of a brigadier, who draws the metal and superintends the distillation generally (and also does the stoking if it be a direct fired furnace) and a different number of smelters called "manœuvres," who discharge and charge the retorts. In the case of direct fired furnaces one of the smelters, known as the "grand manœuvre," works a long shift with the brigadier. Besides the trammers who remove the residues dropped into the cellars, there are generally four boys by day and four by night for every three massives, who supply the smelters with drinking water and perform various light duties such as attending to the prolongs. Gas fired furnaces also have in addition to the smelters a certain number of stokers who attend to the generators. The stokers work 12-hour shifts. The brigadiers generally remain for 24 hours, although in exceptional cases, as at Angleur, they, too, work only 12-hour shifts. The other smelters and the ash trammers are employed only by day, their labor lasting commonly from 5·5 to 6·5 hours in the case of the former and six to seven hours in the case of the latter. At Angleur, where the brigadiers work 12-hour shifts, one of the "manœuvres" per massive remains on duty with the brigadier after the charging is completed and by night there is also a brigadier and a "manœuvre," corresponding to those who are called "metal drawers" in the United States. The maneuver in Belgium generally occupies five hours. In the case of the Rhenish furnaces at Valentin-Cocq and at Flône, which are grouped in massives of 216 retorts, there is a brigadier and crew for each side of the massive. The 54 retorts of the first furnace having been charged, which occupies about 2·5 hours, the 54 of the second furnace are taken in hand, requiring 2·5 hours more. In the meanwhile the brigade attending to the other side of the massive has been working in the same sequence. In this way the maneuver of a single furnace occupies only 2·5 hours, leaving 21·5 hours for the distillation.

Silesian Practice.—With the Silesian furnaces the conduction of the distillation is essentially the same as with the Belgian and Rhenish furnaces equipped with smaller retorts, except that the condensers are not removed for discharging and recharging the retorts (muffles) and spelter is drawn from the condensers only once per 24 hours. The condenser is inserted in the upper part of the muffle, the open end of the muffle under the condenser being closed by a removable clay plate, which is luted in tightly during the distillation. At the beginning of the maneuver this plate is taken away and the residues are raked out through the opening. The muffle is then recharged by throwing ore and coal through the condenser (the lowest section if it be a Dagner or Kleeman condenser). The charger is obliged therefore to put the ore a greater distance than in the case of the Belgian

retorts, and even more deftness is required in filling the Silesian retort than in filling the Belgian. It is hardly possible to fill successfully a Silesian retort longer than 2 m., which necessitates a delivery of about 3 m. through a rectangular opening only 0·15 m. wide and 0·20 m. high. About 45 shovelfuls of charge are required to fill a muffle. The latter is not filled quite to the top, as is done in the case of the Belgian retorts. After the muffle has been charged, the ends of the condensers are closed, and the discharge opening of the muffle having been previously securely luted up, the entire series of operations occupying about six hours, the temperature of the furnace is raised rapidly to whiteness. The volatilization and condensation of the zinc begin after the elimination of the moisture and hydrocarbons of the coal, the same as in the Belgian process, and attain the maximum after about six to eight hours, and remain stationary for another six to eight hours. The activity of the process then diminishes and is concluded in a few hours longer.

In the course of the maneuver the muffles that have become badly defective are replaced, and those which may have been cracked but are still capable of being repaired are patched with clay. In the case of furnaces having two rows of muffles the upper is charged first and then lower. In all the Silesian furnaces the charge for the various retorts is modified as to quantity and character of material according to the position of the retorts in the furnace, but with the general application of gas firing less attention is required in that respect, since the temperature of all the retorts is more nearly alike.

The management of the Rhenish furnaces embodies features of both the Belgian and the Silesian practice. The retorts are discharged and recharged in rows, beginning with the uppermost. The condensers remain in place and the charge is put in through them. Spelter is drawn off only once per 24 hours.

Temperature Maintained.—The reduction and distillation of zinc oxide begins at a moderate degree of heat, but the higher the temperature, the more rapid and complete is the action, wherefore the temperature is maintained in the furnace at a much higher point than what is theoretically required. Speaking generally, the European smelters appear to drive their furnaces harder than the American and obtain relatively a better extraction of metal in the same time; an examination of the analyses of retort residues given elsewhere in this treatise will show this. This better extraction of metal attained by harder driving of the furnaces is not at the expense of the life of the latter, inasmuch as the length of furnace campaign at European smelteries is on the whole probably in excess of the average American practice;

whether it be due to better construction or better management, or both, is immaterial. This conclusion has special reference to the older Kansas-Missouri practice. During the last two or three years in the practice of smelting with natural gas there has been such improvement as to leave but little difference in the results.

The temperature of the distillation furnace attains a maximum toward the end of the distillation period. It falls off greatly when the condensers are taken down and the ashes are pulled out; and sinks further when the cold charge is introduced. The latter cools the retorts to a comparatively low point. The temperature of the furnace is then raised, the charge in the retorts heating up gradually, and finally attaining a temperature within about 100° C. of that of the furnace. The temperature fluctuations during 24 hours are shown in the following determinations by Doctor Otto Mühlhaeuser at an American works (presumably at Lasalle, Ill.):[1]

Time	Temp'ature		Time	Temp'ature		Time	Temp'ature		Time	Temp'ature		Time	Temp'ature	
	R.	F.		R.	F.		R.	F.		R.	F.		R.	F.
10.30	....	1298	3.30	957	1155	8.30	1012	1199	1.30	1078	1243	6.30	1149	1243
11.00	....	1265	4.00	935	1166	9.00	1001	1188	2.00	1078	1221	7.00	1133	1265
11.30	....	1155	4.30	935	1138	9.30	1012	1199	2.30	1078	1232	7.30	1144	1276
12.00	....	1067	5.00	946	1144	10.00	1005	1221	3.00	1100	1221	8.00	1177	1287
12.30	....	1067	5.30	946	1155	10.30	1034	1210	3.30	1111	1232	8.30	1182	1320
1.00	....	1067	6.00	979	1166	11.00	1050	1221	4.00	1111	1232	9.00	1188	1298
1 30	781	1067	6.30	1001	1166	11.30	1034	1210	4.30	1133	1243	9.30	.....	.....
2.00	814	1100	7.00	1001	1177	12.00	1050	1221	5.00	1133	1254	10.00	.....	1309
2.30	869	1100	7.30	990	1177	12.30	1061	1221	5.30	1122	1221	10.45	.....	1188
3.00	924	1111	8.00	1001	1188	1.00	1067	1221	6.00	1122	1210			

The temperatures reported in the above table are in degrees Centigrade. The columns R give the temperature in the retort, and F that of the furnace. The determination at 11 a. m. was made after removal of the condensers; that at 11:30 a. m. after drawing out the residue of the spent charge; that at 12 m. just after putting in the new charge; and that at 10:45 a.m. the next day after withdrawal of the residue. According to Doctor Mühlhaeuser the reduction temperature of zinc oxide is about 1000° C.[2]

Elsewhere the same authority reported determinations of the temperature of the interior of the furnace, near the middle wall, at the end of the furnace where the gas is introduced, as follows:[3]

Time	°C.	Time	°C.	Time	°C.	Time	°C.
10	1122	4	1188	10	1276	4	1315
12	1045	6	1232	12	1287	6.45	1210
2	1150	8	1270	2	1287	8	990

[1] Zts. f. angew. Chem., XVI, xii, 278. [2] Refer to the discussion on p. 198 et seq.
[3] Zts. f. angew. Chem., XVI, xiv, 322.

Saturation of New Retorts.—A new retort inserted into the distillation furnace gives during the first few days almost no zinc at all. A part of the metal distilled in it during this period appears to be sucked into the pores of the retort, combining with the alumina to form a spinel, while a part appears to pass through the pores into the combustion chamber. After the retort has been in the furnace several days and has become partially saturated with zinc and its walls have acquired a sintered texture, it begins to yield zinc, but it is still far from gas tight, the least excess of pressure inside or outside being sufficient to force zinc vapor into the combustion chamber of the furnace or gases of combustion into the retort. The danger of the gases of combustion, surrounding the retorts, finding entrance into the latter, when there is a minus pressure therein, is demonstrated by analyses that have showed a high tenor in nitrogen in the gas issuing from the retorts, which of course could come only from the combustion chamber of the furnace. Obviously these dangers and losses depend a good deal upon the porosity of the retorts. Thus, according to O. Mühlhaeuser, an analysis of an old Rhenish retort showed only 6·42% ZnO, while two from Kansas and Illinois, both made from St. Louis clay, which yields rather porous products, assayed respectively 16·88% and 19·10% ZnO.[1] This subject is discussed further in Chapter XII.

Treatment of Blue Powder.—About 5 to 10% of the zinc distilled and condensed from a charge is collected in the form of blue powder (zinc fume, zinc dust, Zinkstaub, poussière de zinc), but under unfavorable conditions the percentage may easily be higher. An unduly high percentage of blue powder indicates bad smelting practice, since the zinc condensed in that form can as a general thing be made marketable only by resmelting. In resmelting blue powder not only is there that additional product to be handled, but also it occupies room in the retorts which might otherwise be filled with ore, and moreover the zinc of the blue powder being subjected to a second distillation there is necessarily a further loss of metal to be suffered. Blue powder high in metallic zinc and low in zinc oxide and other impurities is used in the arts to a comparatively small extent, especially in connection with the indigo dyeing trade, and some smelters both in Belgium and Upper Silesia prepare a product for that market, but if every one attempted to do so the demand would be far surpassed. In general therefore the zinc smelter is obliged to rework his own make of this troublesome between-product. Because of the film of oxide which coats each minute particle of zinc, it is difficult to make the latter coalesce; a simple heating does not suffice.

[1] Zts. f. angew. Chem., 1902, No. 48.

Montefiore Furnace.—In Europe blue powder was formerly treated for recovery of its zinc by liquating out the metal under pressure in the so-called Montefiore furnace, examples of which are still used at one or two works in Belgium. That furnace had two rows of vertical retorts made in the shape of a boot open at the toe. After the blue powder was charged in, a clay piston attached to an iron rod was inserted into the leg of the boot. Upon heating the retorts with the piston pressing on the charge in their interior, the metallic zinc was melted without suffering oxidation and flowed out through the opening at the toe of the boot. The retorts were 0·73 m. high, 0·183 m. in diameter inside and 0·235 m. in diameter outside. The

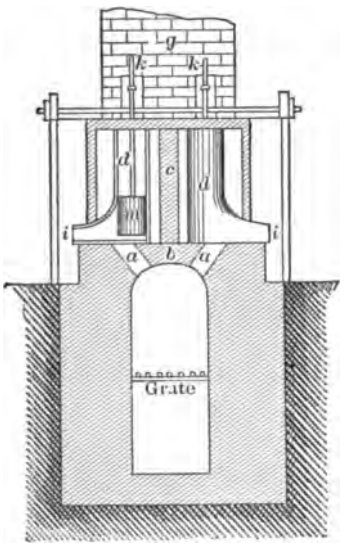


FIG. 357. MONTEFIORE FURNACE.

clay pistons were 0·21 m. high and 0·17 m. in diameter. The grate was 1·73 m. long and 0·47 m. wide. The flames passed through a series of openings 0·08 m. wide in the arch of the fireplace into the combustion chamber, which was 2·04 m. long, 1·05 m. wide and 0·73 m. high. The number of retorts varied from eight to twelve. The charge per retort was about 20 kg. of blue powder. The fusion of the zinc required about three hours firing, after which pressure was applied to the piston and the liquid metal was squeezed out.

In Upper Silesia these furnaces were sometimes attached directly to the ore distillation furnaces, when, of course, the independent grate was omitted.

At Corphalie, in Belgium, a Montefiore furnace reduced from 700 to 900 kg. of blue powder in 12 hours with a consumption of 33 to 44 hectoliters of coal and a yield of 85 to 86% of zinc. Approximately the same yield was attained from blue powder at the works at Lipine in Upper Silesia. The zinc produced by the Montefiore furnace was of poor quality on account of its tenor of oxide, and because of that drawback among other considerations the method of treating blue powder in this manner has been long since abandoned, save at Corphalie and Engis.

Modern Practice.—The general practice nowadays is to mix the blue powder made in the distillation of one day's charge with the ore to be charged on the second day following. The blue powder is not mixed with the entire charge for the furnace, but with only a part of it which is reserved for that purpose. This part of the charge is then introduced into the retorts of a special position in the furnace, usually those of the highest rows if the movement of the gases of combustion be upward, or in the retorts of the tiers nearest the chimney end if the movement of the gases be longitudinal through the combustion chamber. In either case it will be observed that the blue powder part of the charge is put into the retorts which run coolest, because the blue powder will distil at a lower temperature than the zinc of the fresh ore.

If zinc dust be distilled alone a spelter of inferior quality is produced, which must either be mixed with spelter of better grade by remelting or marketed at a discount. On this account the practice of distilling blue powder by itself has been abandoned. Some of the results that were formerly attained in such a distillation are, however, interesting and instructive. At the works of the Nouvelle Montagne Co. at Engis, in Belgium, 43,800 kg. of blue powder assaying 81% Zn was distilled in a 38-day campaign, yielding 33,726 kg. of spelter for a consumption of 912 hectoliters of heating coal and 152 hectoliters of reduction coal. The recovery of zinc was about 95% of the original contents of the blue powder. The loss of 5% must have been almost entirely by failure to condense the volatilized metal, since under the conditions of blue powder distillation but very little zinc ought to be lost in the residue. The daily charge of the furnace was 1200 kg. of blue powder and four hectoliters of coal. Before charging the furnace was cooled down to dull red heat, and the distillation was performed at a low temperature in order to prevent the charge from becoming pasty, which happens when the temperature is too high. The condensed zinc was tapped seven times per 24 hours.[1]

Composition of Blue Powder.—The blue powder produced in distillation

[1] Massart, Revue Universelle des Mines, XXX, 201.

consists of exceedingly minute particles of zinc, each one apparently covered by a thin coating of zinc oxide. It is because of that coating of oxide that the powder cannot be converted into spelter by a simple melting. All attempts in that direction have led to a very impure zinc or a great loss of zinc. The blue powder which is collected in prolongs is of much higher tenor in zinc than the oxidized skimmings which are collected and referred to as "blue powder" at American smelteries, wherein prolongs are rarely used. Ad. Firket has reported the following analyses of blue powder produced in 1898 at several Belgian works, which are of special importance inasmuch as the composition of the ore distilled is also given.[1]

Blue Powder.	A %	B %	C %	F %	H %	I %	I %
Zn	*a* 90·11	*b* 94·04	*a* 91·50	*a* 92·40	*c* 95·50	*d* 88·80	*e* 91·29
Pb	0·82	2·50	0·50	1·70	1·50	1·90	1·98
Fe	0·10	0·30	0·18	....	....	1·32	0·79
Cd	0·005	1·30	0·50	....	....	1·80	0·52
As+Sb	nil	....	0·16	....	....	tr.	tr.
C	3·33	....	?	2·85	2·50	4·10	3·11
Insol		0·50	....	....	....	tr.	tr.
Ore.							
Zn	47·00	46·60	43·00	53·00	50·00	48·50	44·50
Pb	3·80	6·60	1·80	6·50	5·50	6·50	9·30
Fe	10·34	5·50	8·40	?	8·50	8·30	12·50
Cd	0·005	0·06	0·10	?	?	0·18	0·25
S	1·08	1·80	1·50	?	?	2·50	3·40
As	tr.	tr.	0·05	?	?	0·015	0·016
Sb	tr.	nil	0·03	?	?	0·04	0·03
CaO	4·00	2·80	3·50	?	4·00	6·00	4·50
MgO	0·60	0·80	1·20	?			
SiO_2	10·00	13·80	22·00	?	6·00	13·80	13·90

a A small portion in the form of ZnO. *b* Metallic zinc 88·74%; ZnO, 6·60%. *c* Metallic zinc, 85·34%; ZnO, 12·66%. *d* Metallic zinc, 79·16%; ZnO, 11·26%. *e* Metallic zinc, 85·24%; ZnO, 7·54%.

In quantity the blue powder ranged from 2 to 6% of the weight of the spelter produced. In the case of the two columns headed I the ore analyses represent the identical charges from which the blue powder was obtained; in the other cases the ore analyses are the average charge of a year. In general it may be considered that the blue powder produced in Belgium averages 84·5% metallic zinc and 7·5% zinc as ZnO (92% total zinc); 1·6% Pb; 0·8% Cd; 0·04% arsenic and antimony and 3·2% carbon, iron and other substances.[2]

Some analyses of Silesian zinc dust are given by Doctor Victor Steger as follows:[3]

[1] Annales des Mines de Belgique, 1901, VI, 1. [2] Firket, loc. cit.
[3] Sammlung chemischer und chemisch-technischer Vorträge, 1896, I, 11, 80.

	I %	II %	III %	IV %
Zinc	84·463	88·50	90·70	86·60
Zinc oxide	4·888	7·40	3·90	7·70
Cadmium oxide	2·654	1·84	1·83	1·62
Lead	4·276	2·02	2·93	3·55
Ferric oxide	0·903	0·06	0·10	0·09
Lime	2·464	....	....	....
Magnesia	0·239	....	....	....
Insol. residue	0·120	....	....	0·35
Total	100·007	99·82	99·46	99·91

Other Between-Products.—Besides the blue powder proper there are obtained in zinc distillation various other between-products rich in zinc, which have to be reworked. These comprise condenser scrapings, i.e., the crusts of blue powder and oxide which accumulate in the condensers and are scraped out when the latter are taken down during the maneuver; skimmings of blue powder and oxide from the kettles into which the spelter is drawn from the condensers; sweepings from the floor of the furnace house, in which there is likely to be numerous pellets of metal splashed over during the handling of the molten spelter, leaked from condensers, etc.; partially distilled ore removed from broken retorts; and rich residuum reserved from the front ends of the retorts after the distillation is completed. The aggregate quantity of these between-products is very considerable. In Kansas smelteries, where the blue powder proper is not kept separately but is put in with the other material, the combined weight of all is sometimes as much as 40% of the weight of the ore charged, i.e., in smelting 10 tons of new ore there will be perhaps four tons of material assaying 35 to 45% Zn to be returned with it into the furnace.

Besides their own between-products Eastern and European smelters receive considerable supplies of zinkiferous waste products from the process of galvanizing iron.

Length of Furnace Campaigns.—The length of the campaign of which a distillation furnace is capable depends upon the construction of the furnace and the conditions under which it is operated. It is of course obvious that a furnace which is driven hard, i.e., at a high temperature, in order to improve the extraction of metal, cannot be expected to last so long as a furnace which is heated more moderately. It is equally obvious that the more substantially a furnace be built and the better the class of the fire brick, etc., employed in its construction the greater will be its durability. In recent years the tendency in Europe has been toward more substantial construction, and in Belgium nowadays a furnace is expected to stand a cam-

paign of two to three years, and even more, whereas formerly but little more than half that time could be hoped for. In the early days (1844) of the zinc industry in Belgium only two months' regular operation of a furnace was possible.[1] The Silesian furnaces last longer than the Belgian, their average life being from three to four years. At the Hohenlohehütte it is claimed that the average is longer; at the Wilhelminehütte in May, 1893, there was one furnace in course of reconstruction which was said to have been in operation for six years.

As has been remarked elsewhere, it is the practice in Belgium to drive the distillation furnaces harder than in the United States, wherefore the results obtained as to the life of the furnaces in the two countries cannot be properly compared. As a general thing, moreover, the direct fired furnaces of the Belgian type in use in Missouri and Kansas are much inferior in construction to those which are employed in Belgium. The average length of furnace campaign with the direct fired furnaces of Kansas and Missouri is about two years. The records of a number of works in that district from 1896 to 1899 showed the following results:

	Furnace	Months		Furnace	Months
Works A—	No. 1	25	Works D—	No. 1	22
	2	22		2	21
		—		3	16
		Average 23½		4	35
					—
					Average 23½
Works B—	No. 1	26	Works E—	No. 1	32
	2	20		2	30
	3	33		3	25
		—		4	23
		Average 26½		5	23
				6	19
				7	32
				8	14
				9	26
Works C—	No. 1	33		10	26
	2	30		11	34
	3	31		12	34
		—			—
		Average 34½			Average 26½

In computing the above averages the short campaigns made by certain furnaces when it was necessary to put them out of operation because of temporary market conditions have been of course omitted. The condition of the zinc industry in Missouri and Kansas during the decade 1890-1900 was such that furnaces would be put into operation in great numbers when the demand for spelter was strong and the price rising; operations being suspended immediately upon a diminution of demand and a declining mar-

[1] Bulletin de la Société de l'Industrie Minérale, 1888, p. 505.

ket. This practice was the natural result of a great surplus in the smelting capacity as compared with the demand for spelter for consumption. In putting furnaces out of operation of course it would be naturally those in the poorest condition, and giving consequently the poorest results, of which the fires would be drawn. Consequently the averages given above are undoubtedly those of the best furnaces by a process of natural selection.

Relining.—Whenever the fire is drawn from a distillation furnace in all probability that furnace will be relined before putting it into operation again, even if still in a fairly good condition. The extent of the repairs and renewals which will be required, however, will naturally depend upon the condition of the furnace, and will be greater after a long campaign than after a short one. If the fire be drawn from a furnace and the latter be permitted to become cold it is not attempted to heat up the old retorts remaining therein, although they may have been in good working condition before the fire was drawn. For that reason it is frequently the practice to keep a furnace under "dead fire" when it is desirable to discontinue production for only a brief period. In "dead firing" only enough coal will be burned on the grates to maintain the temperature of the retorts at a faint red, or even less. Dead firing becomes very expensive, however, for any but a very brief period, wherefore if the suspension of operation is likely to be long it is preferable to draw the fires completely. In the natural gas fields, on the other hand, where the fuel costs comparatively little, furnaces are sometimes kept under dead fire for considerable periods.

The natural gas fired distillation furnaces of Iola, Kan., and elsewhere have not yet been in use sufficiently long to enable accurate conclusions to be drawn as to their life and the cost of repairs and renewals. The indications are, however, that the more recent and better constructed furnaces will hold out for campaigns approaching three years in duration. Some of the earlier furnaces were very poorly built and lasted only one year. The new furnaces are not only better constructed than their predecessors, but also are better than the old direct fired furnaces, while the present practice is also superior to the old. Hence the longer furnace life which is to be expected.

Tools Required.—The outfit of the principal tools required for a 224-retort direct fired Belgian distillation furnace is as follows: Four scoop shovels; four charge scoops; four fire bars; four rabbles; four spiesses; two stampers; four large retort rabbles; four small rabbles; two ladles; four condenser stands; 10 condenser scratchers; 10 molds; two wire brooms; two straw brooms; two tongs; two bent bars for carrying hot retorts; two straight bars for inserting retorts.

XII.

LOSSES IN DISTILLATION.

The loss of metal in distilling zinc ore varies greatly according to the character of the ore and the conditions of practice. It is seldom less than 10%, and frequently is more than 25%.[1] The loss in zinc distillation is incurred in many ways, which in relative importance may be quite different in various smelteries. These losses are classified as follows: (1) zinc which escapes volatilization; (2) zinc which escapes condensation. The loss suffered in the first way is incurred in the residues which remain in the retorts after the distillation of a charge is completed, those residues containing always a more or less percentage of zinc; the losses which are suffered in the second way are due to (*a*) absorption of zinc by the walls of the retort; (*b*) filtration of zinc vapor through the walls of the retorts into the combustion chamber of the furnace; (*c*) escape of zinc vapor into the combustion chamber through cracks in retorts; (*d*) escape of non-condensed vapor from the retort upon removal of the condenser at the end of the distillation; (*e*) escape of non-condensed zinc from the condensers. The proportion of loss which is experienced by non-volatilization and non-condensation varies considerably in different works. If it were desirable to make a general statement, it might be said that the loss by non-volatilization and that by non-condensation are likely to be each approximately 50% of the total loss; in other words about equal in amount.

Loss of Zinc by Non-Volatilization.—The residues, or ashes, discharged from the retorts always contain zinc, the percentage varying from 2 to 10 or more. In referring to such percentages, it must be borne in mind that the weight of the residues is greater than what is due to the ore alone, since they contain a considerable quantity of ash derived from the reduction coal, besides much unconsumed coal. Inasmuch as the weight of the residues is so largely a factor of the grade of the ore and the percentage

[1] Loss in distillation is sometimes referred to in terms of the ore; thus an ore assaying 50% Zn is said to yield 45%, 5% being lost, which is a loss of 10% of the metal contained in the ore. In this treatise losses are referred to uniformly in terms of the original zinc contents of the ore subjected to distillation.

of reduction coal used in the charge, and the percentage of that coal which is burned, the bare statement of the assay in zinc of any retort residue is of little significance.

The reason why all the zinc in the ore is not expelled during distillation is twofold:

A certain portion of the zinc of the ore charged into the retorts is in combination with sulphur, either as sulphide or sulphate, in which form it is not capable of reduction by carbon and volatilization. The presence of sulphur in the ore at this stage may be due to imperfect roasting or perhaps to some chemical characteristic of the ore which makes it impossible to drive off all the sulphur by roasting. Thus if an ore contain magnesia there is likely to be formed during roasting a certain percentage of magnesium sulphate, which may escape complete desulphurization. When such an ore is subjected to distillation, the magnesium sulphate is decomposed with evolution of sulphurous anhydride, which tends to form zinc sulphide, leading to a loss of zinc. The reactions which take place in the retort are probably exceedingly complicated; they have been described so far as known in a previous chapter, to which reference should be made. Whatever be the nature of those reactions, however, experience has demonstrated that for every unit of sulphur (save that existing as calcium sulphate and analogous compounds) contained in the ore which is charged into the retorts, there will be about two units of zinc held back in the residues.

This is an old rule and a safe one. It is a fact, however, that many retort residues show less zinc held back than the sulphur content of the charge would call for, and there are strong grounds for the theories as to the decomposition of zinc sulphide in the retort, which have been referred to in a previous chapter (vide p. 207).

Besides the loss of zinc entailed by the presence of sulphur, there is likely to be a considerable loss of zinc in the form of oxide, which is due to insufficient heat in some parts of the furnace and perhaps a lessening of the activity of reduction, as the quantity of zinc oxide remaining becomes comparatively small. Professor Prost found in the investigation of a Belgian retort residue assaying 4·76% Zn that only about half of the zinc, specifically 2·36%, was in the form of sulphide.

Irregularity in Zinc Contents of Residuum.—The zinc contents of the retort residues vary greatly, not only in the retorts placed in different parts of the furnace, but also in different parts of each retort. The ashes from the retorts in the cooler parts of the furnace always assay higher than those in the hotter parts, while with respect to any particular retort, the ashes at the front end are richer than those at the rear end. Because of the former

observation, it is universally the practice to charge the upper rows of retorts of an ordinary Belgian furnace, or the retorts nearest the chimney of a furnace which is fired from end to end, with a more easily reducible charge than is put into the other retorts of the furnace; because of the latter observation it is frequently the practice to collect and save the ashes from the front ends of the retorts, when the condensers are removed, and subject them to a redistillation. The retort residues which may assay as low as 1·00 to 1·75 Zn in the inner end of the retorts and only 2 to 2·5% in the middle, may assay as high as 3 to 18% in the first 6 in. of the outer end.

From the foregoing remarks it will be obvious that the sampling of the ashes of the distilled charge of ore is no easy matter. It is not sufficient to reserve ashes of any particular row of retorts, or any number of retorts selected systematically or at random, to determine the percentage of zinc which is contained in them. The only way by which the zinc tenor of the ashes can be determined accurately is to collect the entire quantity discharged from the retorts corresponding to a particular charge and cut them down by quartering or fractional selection in the regulation manner, a rather laborious piece of work, for which few zinc smelters in the United States at least are properly equipped, and consequently is seldom done. Because it is so seldom done, statements as to the zinc tenor of the residues must be taken with a good deal of caution.

The difference in the tenor of the residues remaining in the retorts of various positions in the furnace is illustrated in the experience of the smeltery of the Société Anonyme de la Nouvelle Montagne at Engis, Belgium, as reported by Massart.[1] At those works there were used direct fired Liège furnaces with 70 retorts and direct fired double furnaces with 92 retorts, the latter having 46 retorts per side arranged in six rows. The Liège furnaces were 1·5 m. deep, 2·6 m. wide and 3 m. high. The retorts were 1 m. long, 24 cm. in diameter outside and 19 cm. inside. The daily charge was 1300 kg. of ore assaying 47 to 48% Zn, mixed with 700 kg. of reduction coal. The consumption of heating coal was 2000 kg. The output of spelter was from 470 to 480 kg., the loss of zinc being 18%. The residues assaying more than 6% Zn were enriched by jigging and the concentrate obtained from them was charged into the upper retorts of the distillation furnace, or were worked for zinc white in reverberatory furnaces.

The double furnace with 92 retorts received a day charge of 400 kg. of ore (calamine and blende) assaying 40·32%, together with 72 kg. of blue powder and scrapings, and 160 kg. of reduction coal. On the night shift the furnace, then working hotter, was charged with 500 kg. of ore. The

[1] Revue Universelle des Mines, 1871, XXIX, 313.

retorts were loaded as follows: Lowest row, 11 kg. of ore and 3 kg. of coal; fourth row, 7·6 kg. of ore, 3 kg. of blue powder, etc., and 3·6 kg. of coal; sixth row, 5 kg. of ore, 5 kg. of blue powder, etc., and 5 kg. of coal. The charging of the retorts required 3¼ hours. The residues from the upper retorts contained on the average 9·15% Zn, those of the middle rows 4·67% Zn, and those of the lower rows, 2·28%. The loss of zinc was 11·28%. The retorts in the topmost row lasted 90 days; in the lowest row only six days. The length of furnace campaign was from 150 to 180 days. These results were obtained in the '60s. The short life of the retorts and the furnaces is very striking, but the percentage of zinc recovered indicates good work.

Degree of Distillation Governed by Economical Considerations.—In a subsequent chapter wherein the effect of a varying grade of ore upon the cost of smelting is discussed, it is pointed out how the loss of metal increases as the grade of the ore decreases, which rule follows from the facts that the desulphurization cannot be carried below a certain point, irrespectively of the quantity of sulphur originally present in the ore, whence the loss of zinc in the residues remaining the same for a low grade ore as for a high grade ore and the loss by non-condensation remaining also practically the same, the percentage of loss is obviously greater with reference to the low grade ore than with reference to the high grade ore. Aside from that loss, the degree of loss which will be suffered by non-volatilization is determined largely by economical considerations; that is to say, there will be found a certain minimum to which the zinc can be profitably reduced in 24 hours' distillation. In order to expel an additional percentage of zinc there will be required a disproportionate expenditure of coal, or an increase in the time. Some ores can be distilled more quickly and more completely than others owing to their peculiar chemical composition or physical condition, but the extent to which it is profitable to carry the distillation is governed always by the cost of the process and value of the product.

In Upper Silesia, where the ores distilled have about 20 to 25% Zn, the retort residues should not assay much over 3% with good work, though they often go higher. It is feasible to make them lower than 3%, but unless the price of zinc be very high, it is unprofitable to do so, because beyond that point an extra recovery is attainable only by maintaining a higher temperature in the furnace, or by increasing the proportion of reduction material, or by charging the retorts more lightly, or by all those factors combined and the additional zinc recovered is found to cost more than it is worth. Obviously the extent to which it is profitable to sweat out the last of the zinc in a charge is governed considerably by the market

price of the metal. At the Hohenlohehütte in distilling an ore which averages 20% Zn, of which 75% is recovered, the residues assay 3 to 5% Zn. A sample assaying 5% Zn contained also 5·48% S, of which 1·1% was in the form of $CaSO_4$, and the remainder as sulphides. In Belgium, Rhenish Prussia and Westphalia, where a higher grade of ore is distilled than in Upper Silesia, the ashes may assay 4 or 5% Zn and the work is considered well done.

Ad. Firket, in the paper which has been quoted frequently in previous sections of this treatise, gives the following average analyses of retort residues produced by Belgian works in 1898, the designating letters corresponding to those in the table of average charges which will be found on p. 502, Chapter XI.

Works	A %	B %	C %	D %	E %	F %	*a*G %	*a* G %	H %	*a*I %	*a* I %
Zn.....	4·00	2·50	3 to 7	2 to 3	?	?	2·60	3·94	4·00	3.40	4·20
Pb.....	5·00	8·50	1·26	9·00	11·50	9 to 12	2·13	11·60	10·00	8.10	9·50
Fe.....	16·55	14·50	?	?	?	15·65	15·72	11·19	?	?	?
Ag.....	0·016	?	?	?	?	0·017	....	0·035	0·01	0.05	0·085
Cu.....	0·05	?	?	?	?	?	....	0·19	....	?	?
Cd.....	nil	?	?	?	?	?	....	....	....	?	?
As.....	nil	?	?	?	?	?	....	....	....	?	?
Sb.....	nil	?	?	?	?	?	....	0·25	....	?	?
S.....	?	4·00	?	?	?	?	0.62	1·72	?	2·10	3·50
CaO....	2·50	2·50	?	?	?	7·10	4.00	1·33	8·00	?	?
MgO....	0·45	1·50	?	?	?	1·20	2.16	0·73	?	?	?
SiO_2....	)		?	?	?	)	34.30	32·23	17·50	?	?
C.....	50·00	60·00	?	?	?	41·60	30.02	30·40	?	?	?
Al_2O_3..	)		?	?	?	)	2.60	1·10	?	?	?

a Residuum from different mixtures of ore.

Firket considers that the average of the residues drawn from the retorts in Belgium in 1898 would assay 4·14% Zn, 5·55% Pb and 0·0209% Ag. The weight of the residuum varies between 65 and 70% of the ore charged, i.e., the ore proper plus the between-products mixed with it. A charge of 1000 kg., composed of 460 kg. of roasted blende, 410 kg. of calcined calamine and 130 kg. of skimmings, condenser scrapings and refuse from the sifting of blue powder, of which the tenor varied between 40 and 50% Zn, 6 and 7% Pb, 0·004 and 0·005% Ag, 15 and 18% Fe and 1 and 1·5% S, was distilled at a certain Belgian works after admixture with 400 kg. of lean coal assaying 9% ash, 8·5% volatile matter and 82·5% fixed carbon. The residuum weighed 680 kg. It contained 4% ZnS, 0·09% $ZnSO_4$, 3·10% metallic lead, 0·006% Ag (carried by the lead), 1·91% PbO, traces of PbS, 5·69% metallic iron (more or less carbureted), 2·48% FeS, 15·57% oxides of iron (FeO, Fe_3O_4 and Fe_2O_3) and 18·07% of carbon, besides silica, alumina, lime and magnesia. The ferrous oxide, lead oxide and the earthy bases were combined with silica as silicates. A remarkable

feature of this interesting analysis is the absence of ZnO, more or less of which usually escapes reduction.

In the United States where the ore regularly distilled assays from 60 to 70% Zn, there are few works which make residues assaying lower than 7% Zn. With such ore in Kansas and Missouri the residues from the direct fired furnaces used to assay 6 to 18% Zn, averaging perhaps about 10 to 12%.[1] A charge composed of 70% roasted blende and 30% raw calamine (hemimorphite), the mixture averaging 58·03% zinc and 1·036% S, distilled with 2275 lb. of coal and coke (45·5% of the ore) which itself contained 1·9% S, produced 2335 lb. of residue assaying 7·12% Zn. The residue retained therefore 166·25 lb. of zinc, or 5·73% of the zinc in the ore charged. This was quite satisfactory work, especially in view of the fact that the ore and coal together contained 95·03 lb. of sulphur, which should have theoretically held back 194·22 lb. of zinc. At the Lehigh Zinc Works, at South Bethlehem, Penn., in 1894, the retort residues from the distillation of an ore averaging 48 to 50% Zn assayed 6 to 7% Zn. In considering the assay of retort residues, it should be borne in mind that 7% Zn in the residues from a 70% ore indicate a much smaller loss than 7% Zn in the residues of a 50% ore, because in the latter case the weight of the residues from a ton of ore is considerably greater than in the former case.

Scorification of Zinc Oxide.—A very small part of the zinc which is lost in the retort residues is due to scorfication of zinc oxide, which is taken into the slag formed by the fusible components of the gangue and coal ashes, notwithstanding the fact that zinc silicate is reducible by carbon. However, the percentage of zinc in such slags appears to be small, and there is reason to believe that part of it exists in the form of undecomposed sulphide entrained by the slag. The composition of a zinc retort slag is shown by the following analysis, for which I am indebted to Mr. Otto Rissmann, manager of the Cherokee-Lanyon Spelter Co.: SiO_2, 64·92%; FeO, 9·93% (Fe, 7·72%); Al_2O_3, 10·08%; CaO, 6·17%; MgO, 1·84%; Zn, 2·40%; S, 0·91%; Na_2O, 3·44%. Mr. Rissmann calculates all the sulphur as being combined with zinc, giving 2·76% ZnS and leaving zinc enough to make only 0·68% ZnO, on which basis the total of the analysis foots up to 99·82%. The sample assayed was slag from an old retort at a works at Pittsburg, Kan., in which a mixture of Joplin blende and silicate had been smelted. The presence of soda in the slag was attributed partly to the use of a salt glaze inside the retorts. Mr. Rissmann reports other analyses showing less iron (4·50 to 6·25%) and more zinc (3 to 7·42%).

The analysis quoted above indicates that with an ore of which the gangue

[1] Since 1900 the gas smelters have succeeded in obtaining a better extraction.

is very silicious and low in iron, as is the case with the Joplin ore, the slag that will be formed is too silicious and non-ferruginous to absorb much zinc oxide, and although with retorts which are made of a batch containing free bases (after burning) there may be corrosion of their walls, with retorts of a silicious character the corrosion from such a slag would not be of any great consequence.

Recovery of Values from Retort Residues.—It has been proposed to concentrate mechanically the residues from the retorts to recover part of the zinc contained therein, chiefly in the form of undecomposed sulphide. It would appear to be an easy matter to make a high grade concentrate from residues assaying 5 or 6% Zn, especially in view of the fact that crude zinc ores of lower grade are dressed very profitably. Owing to the physical condition of the retort residues, however, the particles of the ore being attached to pieces of slag or partially glazed over therewith, the recovery of zinc from them by gravity separation has never been a very successful undertaking. I am unaware that retort residues are being thus treated anywhere in the world at the present time for the recovery of their zinc.

It is, however, the practice at a large number of works in Belgium, and elsewhere in Europe, whereat ferruginous and plumbiferous ores are smelted, to concentrate the residues for lead, and occasionally for iron. The residues are crushed and screened, and then jigged, the iron being perhaps removed by magnetic separators before jigging. The lead concentrate is briqueted or sintered, and then smelted with other ores in blast furnaces. Firket gives the following analyses of concentrated products obtained from crude residues of which the analyses have been given in a previous section of this treatise, the reference letters being the same in each case:

	A		B	C	D	E		F	G	H	I
	1st class %	2nd class %	%	%	%	1st class %	2nd class %	%	%	%	%
Zn...	3·87	5·00	6 to 7	3 to 15	...	5·42	5·30	.....	6·89 to 7·70	8·0 to 12·0	2·50 to 7·00
Pb...	24·25	13·16	40 to 45	4 to 30	30·00	19·10	15·25	31·90	35·00 to 57·00	30·0 to 35·0	10·00 to 48·00
Ag...	0·032	0·016	.....	.....	.....	0·083	0·019	0·041	0·04 to 0·07	0·04 to 0·05	0·049 to 0·16
Fe...	42·75	20·68	15 to 20	.....	.....	15·13	32·25	26·18	11·26 to 21·22	.....	11·77 to 24·03
SiO_2.	18·66	44·67	15 to 20	.....	.....	13·52	21·29	28·20	13·56 to 15·38	.....	12·75 to 60·00

The idea of mechanical concentration has been attractive because of the possibility of regaining not only the lead and the undistilled zinc compounds, but also the large quantity of unburned coal which exists in the residuum. Of course only that part of the coal charged into the retorts is consumed which can obtain oxygen from the ore—not a large percentage

of the whole. Doctor Steger made recently a careful investigation of this question in Upper Silesia. He made two distillation tests with the following results:

Number of Test	Roasted Blende Under 2 mm. Size		White Calamine (a)		Red Dolomitic Calamine (a)		Av. Zinc Tenor of Charge	Reduction Coal			Loss of Zinc in Distillation	Zinc Tenor of Residues
	Am't	Zinc Tenor	Am't	Zinc Tenor	Am't	Zinc Tenor		Coke 4-30 mm.	Coal 1-15 mm.	Total		
	%	%	%	%	%	%	%	%	%	%	%	%
I.....	29·1	34·0	23·6	14·0	47·3	9·5	16·29	28·4	17·7	46·1	29·49	3·20
II.....	21·0	34·0	26·3	14·0	52·7	9·5	15·41	26·0	12·2	38·2	28·06	4·12

a The calamine was dried, but not calcined.

Six analyses of muffle residues were as follows: I, 3·20% Zn; II, 4·12%; III, 4·01%; IV, 5·41%; V, 5·16%; VI, 5·36%. In washing V and VI concentrates assaying 13·80 and 14·60% Zn, respectively, were obtained, but only 7·92 and 8·15%, respectively, were in the form of ZnO. Ashes amounting to 7063 kg. from these experiments were washed for unburned coal, yielding 1505 kg. of 4 to 30 mm. size, at a cost of 12·71 marks, or 0·845 mark per 100 kg., which was too dear. The conclusion reached by Doctor Steger was that it would be unprofitable to dress the Silesian muffle residues, and the hope that the zinc in the old heaps would be available is not likely to be realized.

At those works where the residues are jigged for the recovery of lead the washed coal is utilized in exceptional cases. At Monteponi, Sardinia, it is employed for the calcination of calamine; at certain Belgian works it is given away.

Loss of Zinc by Non-Condensation.—The loss of zinc by failure to condense, in the form of spelter and blue powder, that which has been reduced and volatilized occurs through absorption by the retorts, filtration through the retorts, breakage of the retorts, escape of uncondensed zinc at beginning of the maneuver, and by the escape of zinc vapor from the condensers throughout the period of distillation.

Absorption by Retorts.—A new retort inserted in a furnace does not begin to make its maximum output of zinc until several days after it has been placed in position. This is due largely to absorption of metal by the clay, with which it combines as a zinc aluminate; i.e., an artificial zinc spinel. It is that compound which gives the deep purplish blue coloration to old zinc retorts. Such old retorts will be found upon analysis to contain 6% Zn or more. F. C. Degenhardt found that old retorts at the works at South

Bethlehem, Pa., sometimes had as much as 21·47% ZnO, a series of analyses giving the following results: 21·47%; 18·09%; 15·72%; 10·72%; 6·10%; the first was dark blue; the last was nearly white.[1] In 1894 an old retort at the same works showed 19·34% Zn; its weight when new was 103 lb., and it gained 22·9 lb. At Lipine, Upper Silesia, old retorts are said to assay 8% Zn. In Kansas and Missouri they contain usually from 6 to 15%.

The loss of zinc by absorption in the retorts is no inconsiderable item in the total loss; for example, the retorts 0·15×0·34×1·45 m., inside measurement, used at a certain works in Rhenish Prussia, weighed 70 kg. when burned. If an old retort should assay 6% Zn it would hold about 4·2 kg. of metal. The works in question consume 12,000 retorts per annum, which would mean a loss of approximately 50·4 metric tons of zinc by absorption or 0·63% on the output of 8000 tons of spelter. An American works distilling a charge assaying 60% Zn in retorts weighing 140 lb. (after burning) of which the breakage is 2·5% daily loses 1·12% of the zinc in the ore distilled by absorption in the retorts, which assay 8% Zn when thrown away; this estimate of loss is based on the weight of the new retorts, whereas if it were reckoned on the weight of the old material to which the assay (8% Zn) refers it would come to about 1·20%.

There is reason to believe that the loss of zinc by absorption in the retorts is greater than is indicated by the above analyses, at least in Upper Silesia, according to an investigation by the late Edmund Jensch, inspector at the Kunigundehütte, which bears every evidence of reliability. He reported five analyses of old retorts with the following results:[2]

	I	II	III	IV	V
SiO_2	50·20%	48·64%	46·50%	52·14%	49·75%
Fe_2O_3	0·61	0·73	1·21	0·75	0·96
Al_2O_3	30·80	33·58	36·84	28·59	31·82
CaO	0·48	0·35	0·60	0·40	0·52
MgO	0·56	0·39	0·41	0·35	0·38
ZnO	17·64	16·38	14·11	18·21	16·63
CdO	0·005	0·071	0·08	0·01	0·13
Alkalies	0·21	0·300	0·25	.....	0·10
Total	100·505	100·301	100·30	100·45	100·29

Analyses I and II were of samples from 50,000 kg. of old retorts from the Kunigundehütte, near Kattowitz, and the Clarahütte and Franzhütte, at Schwientochlowitz, ground to pass a 4 mm. screen. The material had previously been cleaned of slag and parts high in spinel and was in the form usually employed as old material in the preparation of the batch for retort

[1] American Chemist, 1875, p. 355.

[2] Sammlung chemischer und chemisch-technischer Vorträge, III, vi, 215.

manufacture. Analyses III, IV and V were from the Beuthener, Rosamunde and Godulla works respectively. An average of the five analyses shows 0·059% CdO=0·052% Cd. An average of seven analyses, including the five above specified and two others which returned 13·18% ZnO and 15·96% ZnO, came to 16·02% ZnO=12·87% Zn.

According to the statistics of the Oberschlesischen Berg- und Hüttenmännischen Vereins the consumption of muffles in 1895, 1896 and 1897 was respectively 169,218, 174,512 and 179,299, an average of 171,010. Assuming the weight of a dried and burned muffle to be 260 kg. the weight of 171,010 muffles would be 44,462,600 kg. and an average assay of 0·052% Cd and 12·87% Zn would imply a loss of 23,120 kg. of cadmium and 5,722,-357 kg. of zinc by absorption in muffles.[1] According to the same statistics the production of spelter in Upper Silesia in 1895, 1896 and 1897 was respectively 95,430, 98,323 and 95,547 metric tons, an average of 96,433. Assuming that 70% of the zinc in the ore was recovered by smelting, the original tenor of the ore would have been 96,433÷0·70=137,757 metric tons and the loss by absorption in the muffles 5,722·357÷137,757=4·15%, a figure which seems incredible.[2] Yet admitting that Jensch's average of the zinc contents of the old muffles may be too high, and instead of an assay of 12·87% there is only half that quantity, the loss of zinc by absorption in Upper Silesia would still be 2·08%; this is nearly 10% of the total loss in distillation in the average Silesian practice.[3]

It will be observed that the relative loss by absorption in the retorts increases as the grade of the ore decreases. The assay of the old material appears to be about the same irrespective of the grade of the ore distilled, wherefore the quantity of zinc held back is proportionately greater with a low grade ore than with a high grade. For example, if in distilling 25,000 lb. of ore of various grades there were obtained in each case 2250 lb. of old retort material assaying 6% Zn, there would be held back 135 lb. of zinc and if the ores contained respectively 60%, 40% and 20% Zn the losses would be as follows:

25,000 × 0·60 = 15,000;	and 135 ÷ 15,000 = 0·90%
25,000 × 0·40 = 10,000;	and 135 ÷ 10,000 = 1·35%
25,000 × 0·20 = 5,000;	and 135 ÷ 5,000 = 2·70%

[1] This is calculated on the weight of new muffles, while the assays are of old muffles which have gained largely in weight. If the same assays were referred to the weight of the old muffles the tenor of zinc and cadmium in them would be correspondingly greater.

[2] Besides the factors employed in the above calculation, which affect the result, there are others which should be taken into consideration, i. e., the percentage of zinc in the spelter produced and the quantity of zinc produced and sold as blue powder, but such minor factors would not materially affect the result.

[3] Compare with data on p. 543.

This same principle appears to be true with respect to other items of loss in distillation; it is discussed at more length in Chapter XV.

The absorption of zinc by the retorts takes place chiefly during the first week of their use, during which period the new retort produces comparatively little spelter. This fact, together with the losses of metal that are experienced in other ways, has been shown by Doctor Otto Mühlhaeuser in careful experiments at an American works, using retorts made of St. Louis clay, his results being as follows:

Day	Block Zinc	Between-Products				Total Zinc in Btw. Prod.	Losses in				Total losses
		Drop Zinc	Zinc Dust	Skimmings	Condenser Scrapings		Ashes	Volatilisation thro' Condensers	Volatilisation thro' Retort Walls	Absorption	
	%	%	%	%	%	%	%	%	%	%	%
1	29·52	18·90	13·39	3·40	2·39	38·08	2·77	1·52	2·09	25·97	32·35
2	30·85	20·14	15·84	2·66	0·66	39·30	2·19	1·73	2·19	23·78	29·89
3	23·89	31·40	17·60	4·33	1·79	55·12	1·90	1·96	2·74	14·37	20·97
4	36·08	24·68	17·60	3·81	1·80	47·89	2·25	1·87	3·18	8·77	16·07
5	36·96	22·37	17·83	3·46	2·40	46·06	3·69	1·04	3·00	9·24	16·97
6	50·33	22·42	9·67	3·55	0·60	36·24	3·15	1·09	2·36	6·81	13·41
7	53·21	21·44	10·90	2·54	3·60	38·48	3·46	1·07	2·07	1·70	8·30

A retort that had been in use for eight days weighed 163 lb. and assayed 13·5% Zn. Another one, used for 20 days, weighed 171·25 lb. and assayed 12·75% Zn. Still another, used for 135 days, weighed 176·5 lb. and assayed 15·15% Zn. The degree of penetrability of zinc vapor into the walls of the retorts depends upon the nature and number of the cracks which form in them and the thickness and fluidity of the glaze which forms on them. The greater the number of cracks, the deeper they extend into the walls, and the thinner the glaze, the more easily will the zinc vapor penetrate into the walls and the greater will be the loss of metal by absorption and filtration in and through them. The resistance which a retort presents against such penetration is one of the chief factors in determining its value, since thereby the recovery of metal is greatly affected.[1]

Filtration Through the Retorts.—The loss of zinc by filtration through the walls of the retorts is due to the porosity of the latter, and consequently to the inferiority of their manufacture. Given a retort which is more or less permeable, the loss which will be experienced in this manner is largely a function of the difference between the pressure of the vapor inside of the retort and that of the gases of combustion on the outside. That difference is greatest in those furnaces, especially the direct fired, in which there is necessarily a low pressure in the combustion chamber because of the draught of the chimney. With gas fired furnaces, on the other hand, in which there

[1] Zts. f. angew. Chem., XVI, xii, 278 and 282.

is a plenum of pressure in the combustion chamber, it is more difficult for zinc to escape. The extent to which loss of zinc may be suffered by filtration through the retorts is not easily measured, but the practical result of a higher saving with gas firing is due, no doubt, partly to a reduction of the loss in that manner. The loss of zinc by filtration is also reduced by glazing the retorts outside and inside, either by a preliminary treatment with a fusible coating or by the natural glaze which is formed in the furnace. The loss of zinc by filtration has been reduced most effectively, however, by the manufacture of retorts under great hydraulic pressure with the addition of a certain percentage of coke dust in the batch, whereby the walls are made very dense.

Breakage of Retorts.—The escape of zinc vapor from the retorts through cracks, whether the latter be present originally in the retorts or be formed after the retorts have been inserted in the furnace, is a very important source of loss, since such cracks afford a direct vent into the combustion chamber and zinc may be lost through them for a long time before discovery is made. With direct fired furnaces which are operated by means of a chimney draught, the loss of zinc is made known by the appearance of white smoke from the chimneys, especially during the intervals between stoking. At other times the white smoke may be hidden in the dense black smoke of the coal. With gas fired furnaces the pressure in the combustion chamber may be so high that a crack in a retort will not permit the escape of vapor from the latter, but on the contrary gas from the combustion chamber will enter the retort, in which case zinc vapor is oxidized and will be carried away through the condenser. As soon as a retort is discovered to be broken, the condenser is removed and its charge is withdrawn, to be mixed with blue powder and other between-products and redistilled another day. The presence of a broken retort is manifested by the appearance of the flame at the nose of the condenser. With direct fired furnaces and those in which there is a suction in the combustion chamber, the condenser flame dies out, because the gas from the retort is no longer being forced out only through the condenser by its own pressure, but is being drawn into the combustion chamber and thence up the chimney. With such furnaces as have a plenum of pressure in the combustion chamber, the appearance of the condenser flames will depend upon the relative pressure inside and outside of the retort. If the external pressure be the higher, a broken retort is discovered by an elongation of the condenser flame, which also becomes somewhat brownish red instead of bright bluish green. The experienced smelter by observation of his condenser flames discovers a broken retort generally before the zinc smoke shows itself clearly at the chimney, but not always. In any case

there will probably be a considerable loss of zinc before discovery is made. The extent of that loss will depend largely upon the percentage of retorts which are broken.

The breakage of retorts at Angleur, Belgium, is about 2% of the whole number daily; at other Belgian works, where more corrosive ore is distilled, it may amount to 3%. The Rhenish and Westphalian works experience about the same breakages. At Münsterbusch (Stolberg) it is 2 to 2·5%. At Dortmund, in 1894, it was said to be about 1·7%; at Gladbach, 1·7 to 2·2%; at Hamborn, 3·2. The retorts are made by hydraulic pressure at all the works mentioned, except Angleur where they are bored out. Probably a breakage of about 2·5% is a fair average in both Belgium and Rhenish Prussia; this is equivalent to an average life of 40 days.[1]

In the Western districts of the United States the average breakage is probably between 2 and 3%, or about the same as in Belgium. The retorts are inferior in their manufacture, but the ore distilled in them is more docile. In distilling the New Jersey ore, which is high in iron and manganese, comparatively short lives and high percentages of breakage are experienced.

The figures cited above are averages. The life of the retorts varies of course in different parts of the furnace, those in the hottest places lasting for less time than those which occupy the cooler. In gas fired furnaces, wherein a more equable temperature is maintained, the difference is less than in direct fired furnaces.

The daily breakage of muffles in Upper Silesia ranges generally from 2 to 4%; probably 2·5 to 3% is a fair average. The statistics of all the works in Upper Silesia for 10 years are given in the following table:

Year	No. of Muffles in furnaces	No. of Muffles consumed	No. of times replaced	Average life, days	Average daily breakage, %
1891.....	17,680	159,285	9·01	40·0	2·50
1892.....	17,960	156,147	8·69	41·4	2·41
1893.....	18,506	170,350	9·21	39·1	2·56
1894.....	18,664	170,941	9·16	39·3	2·55
1895.....	18,656	169,218	9·07	39·7	2·52
1896.....	18,691	174,512	9·27	38·8	2·58
1897.....	18,831	179,299	9·52	37·8	2·65
1898.....	20,591	210,575	10·22	35·2	2·84
1899.....	21,669	229,305	10·59	34·0	2·94
1900.....	23,604	248,894	10·55	34·1	2·92

The breakage of retorts occurs partly because of cracks which develop under the influence of heat and partly through their corrosion by slags,

[1] Dividing 100 by the average percentage of breakage, gives the average life in days; similarly dividing 100 by the average life in days, gives the average daily breaking in per cent.

and partly by exposure to intense heat, whereby the clay is simply fused. If the retort be hand-made, or molded by means of an augur machine, it should be examined carefully for any signs of cracks, because if there be any such they will be sure to develop in the furnace. Such retorts should be rejected. A works in Kansas increased its saving considerably by care in this respect. With retorts made under hydraulic pressure there is much less likelihood of originally defective vessels. It was formerly the practice to patch defective retorts with clay from the inside, this being a regular part of the maneuver. That practice is no longer continued except perhaps in Upper Silesia. In the United States a retort which is cracked is promptly discarded.

The action of corrosive slags on the retorts eats small holes through them, which may permit the escape of a good deal of zinc before they are large enough to be evident. With direct fired furnaces, the retorts of the lowest rows, being directly over the fireplace, are exposed to a very much higher temperature than the upper rows. By raising the temperature sufficiently to drive off the zinc from the upper rows, consequently, it is very likely that the heat will be far too high for some of the retorts in the lowest rows to withstand, and if such retorts have been previously weakened by the corrosion of slag from the inside, they are likely to develop large holes. Such retorts are said to have been "butchered." A good example of a butchered retort is shown in the accompanying engraving from a photograph taken at a smeltery in Kansas. Retorts may also be butchered in gas fired furnaces either with natural gas or with producer gas firing.

The most severe trial of the retorts comes during the night, when the furnace is being driven hardest, and especially toward the end of the distillation, when the charge is being pinched to get the last of its zinc and when the endothermic reaction which takes place within the retorts has diminished in activity and less heat is being absorbed in its performance.

Escape of Uncondensed Zinc at Beginning of the Maneuver.—When the condensers are removed at the end of the distillation, the retort which at that time is less than half filled by the residue of the charge, contains a certain quantity of zinc vapor which has had not sufficient tension to be forced over into the condenser. There will be probably a large percentage of unconsumed carbon in the residue, but the zinc oxide having been mostly reduced, there is no longer any agent to combine with the carbon and maintain the evolution of carbon monoxide gas. Consequently the retort remains filled with gas which is a mixture of carbon monoxide and zinc vapor. When the retort is opened and its contents thereby exposed to air, this vapor is oxidized and lost in the form of zinc oxide. Various devices have been

FIG. 358. A BUTCHERED RETORT, PHOTOGRAPHED AT PITTSBURG, KAN.

proposed to suck out and recover the vapor remaining in the retort after completion of the distillation, among others by B. Kosmann[1] and C. Palm.[2] None of these devices are in use, however, at the present time, whence the inference is that they were inefficient; or at least unprofitable. The loss of zinc suffered in this way can hardly be of much importance.

Escape of Zinc Vapor from the Condensers.—The escape of zinc vapor from the condenser is the most important loss which falls under the caption of non-condensation. In order to obviate it entirely it would be necessary to use condensers which would be too clumsy for practical purposes. With the comparatively short condensers that are employed it is necessary to regulate the temperature very closely in order to reduce the loss to a minimum. Even if that be done, the metallurgist is confronted by the two alternatives of running with condensers too cold and producing a high percentage of blue powder, which will have to be redistilled, thus detracting from the capacity of the furnace, and being moreover subjected to further loss by the second distillation; or running with condensers too hot, when a part of the vapor will fail to be liquified and will burn at the end of the condensers with the brilliant greenish white flame characteristic of zinc. Nor can the condensers be regulated without consideration of the distillation in the retorts; for example, if the retorts be subjected to an excessively high temperature in order to pinch out the last of the zinc in the charge, the condensers are apt to be too hot and the extra loss of uncondensed zinc escaping from them may offset the increased volatilization from the charge. With the ordinary Belgian furnace, therefore, experience shows that it is not economical to drive out the charge beyond a certain point, aside from the extra consumption of coal.

The regulation of the temperature of the condensers is no easy matter, inasmuch as the conditions are bound to vary in different parts of the furnace because of the necessary method of charging. Immediately after a retort is charged it becomes cold on account of the heat absorbed in raising the temperature of the charge. While the charge is being gradually raised to the temperature of the furnace, heat is absorbed in driving off the water contained in it and the volatile hydrocarbons of the coal. With the active development of carbon monoxide the condensers become strongly heated, and after the zinc distillation is well under way the temperature in the condensers attains its maximum, not only because of the sensible heat of the gases issuing from the retorts but also because of the latent heat liberated

[1] German patent. No. 5923; Berg-u. Hüttenm. Ztg., 1879, p. 280; Chem. Ztg., 1879, p. 9 and p. 491; Dingler's Polytech. Journ., CCXXXV, 281, and CCXXXVI, 250.

[2] German patents, Nos. 15,116, 16,046 and 16,305; Wagner's Jahresberichte, 1883, p. 171.

by the condensation of the zinc vapor. Toward the end of the distillation the temperature of the condensers falls off.

It may happen that in attempting to run the condensers at the minimum temperature the latter will be allowed to sink to the point where the molten zinc accumulated in their bellies will solidify, necessitating their removal and the substitution of fresh ones. This calamity being feared by the smelters more than the loss of uncondensed vapor the probability is that the condensers will be generally run too hot rather than too cold. It is aimed generally to maintain the temperature of the condensers at about 500° C., which is 85° above the melting point of the metal. In Silesian furnaces equipped with the Dagner and Kleeman condensers, which are entirely contained within the niches, the temperature of the latter can be regulated by means of the counterweighted doors which close the front of the niches. Not so with the ordinary Belgian furnaces, in which the condensers and their prolongs project beyond the façade of the furnace. An arrangement of furnace front whereby the temperature of the condensers can be controlled in such cases has recently been patented by George G. Convers and Arthur B. De Saulles, of South Bethlehem, Penn.[1]

The loss of zinc fume through the condensers is reduced by the use of prolongs, or "ballons," which in their simplest form are merely a cone of sheet iron, that is fitted over the nose of the condenser. A more common form is a long sheet iron canister, with a hole in the outer end for egress of the gas. In prolonging the course which the gas has to travel and subjecting it to the stronger cooling effect of the air further from the façade of the furnace, circulating around the iron of the prolongs which is a far better conductor of heat than the clay of the condensers proper, a large percentage of the zinc in the vapor which would otherwise escape, is condensed in the form of blue powder, or dust. The carbon monoxide escaping from the end of the prolong may be ignited, or may not be. If it be allowed to escape unburned, arrangements should be made to remove it from the furnace house, because it is highly poisonous. With some kinds of prolongs there is considerable danger from explosion of the gas by air drawn in through leaks and the opening which must almost always be provided for the insertion of a rod to remove obstructions; this danger is the more likely in vertical prolongs (not used with Belgian and Rhenish furnaces) in which the column of ascending hot gas creates a draught, that may draw in air, but of course there will be no explosion unless the temperature inside of the prolong be at, or higher than, the ignition point of the mixture of

[1] United States patent, No. 694,137, Feb. 25, 1902.

gas and air.[1] These drawbacks to the use of prolongs, together with the increased attention which they demand, lead to a considerable difference in opinion as to the advantage to be derived from them. In Upper Silesia they find extensive employment, except in such works as have adopted the Kleeman and Dagner condensers, the gas issuing from which is burned and the zinc fume recovered as oxide. In Belgium and Rhenish Prussia they are used generally and it is considered that they lead to an extra saving of about 2% of the metal in the ore distilled. In the United States, on the other hand, such works as have at times used prolongs have now generally abandoned them on the ground that the saving effected did not compensate for the disadvantages. At a certain works in Kansas the use of prolongs led to an extra recovery of 0·5 to 1 lb. of zinc per retort, the mean between those extremes corresponding to 168 lb. per furnace of 224 retorts. Assuming that the furnace was charged with 10,000 lb. of ore assaying 66% Zn, the increased recovery would have amounted to 2·55% of the zinc in the ore. Notwithstanding that apparently important extra recovery, it was considered that there was no ultimate economical gain. It must not be forgotten in this connection that the zinc recovered from prolongs is in the form of blue powder, which has to be redistilled, wherefore the extra recovery of metal is not clear gain, or else after sifting must be marketed as zinc dust.

Improved Condensers.—Until a successful furnace in which the retorts will discharge their vapor into a common condensing chamber is invented, the ideal condenser is one which will permit a close regulation of the temperature and afford a free escape for the carbon monoxide gas without admitting any air. These are, however, somewhat difficult conditions to fulfil, inasmuch as the volume of gas to be discharged varies greatly at different stages of the distillation, and no chances should be taken on throttling the gas stream so as to create an improperly high tension. The only practical method of preventing oxidation of zinc vapor in the condenser is to maintain such a stream of carbon monoxide issuing constantly from the condenser that no air can find entrance therein.

The Silesian type of furnace permits of arrangements for the condensation of the zinc vapor which are not possible with the Belgian type of fur-

[1] According to Roberts-Austen, carbon monoxide does not ignite until dull red heat (Introduction to the Study of Metallurgy, p. 171). Dull red heat corresponds to a temperature of 550° to 625° C. Mallard and Le Chatelier determined the ignition temperature of a mixture of 70% air and 30% hydrogen to be 552° C.; this temperature is raised by the introduction of carbon monoxide, of which the ignition point is higher than that of hydrogen; with a mixture of 70% air and 30% carbon monoxide, the kindling temperature varied between 650° and 657° C., while a larger proportion of carbon monoxide increased it to 725° C. (Steinmann, Compendium der Gasfeuerung, p. 10).

nace and are rather difficult with the Rhenish type. These arrangements comprise a long list of special forms of prolongs, or "ballons," and the condensers known by the names of Kleeman and Dagner. The sheet iron ballons which are described subsequently in this chapter are now used in Upper Silesia only by those works which make a practice of recovering zinc dust, which is to be prepared as a marketable product. The other works in Upper Silesia use commonly either the Kleeman or the Dagner condenser.

Kleeman Condenser.—The Kleeman condenser[1] consists of a rectangular clay pipe, near the outer end of which is attached a short piece standing vertically, so that the whole forms an L with unequal legs, as shown in the accompanying engraving. The vertical member which is about 0·1 m. long supports at its upper end a cast iron grate, on which an incandescent bed of coke or cinder, perhaps 0·1 m. thick, is maintained. The inner end of

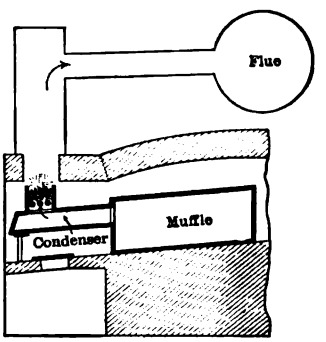


FIG. 359. KLEEMAN CONDENSER.

the condenser rests in the muffle in the ordinary manner. The outer end is closed by a rectangular iron plate, the joint being tightly luted with clay. In the center of the plate there is a small round hole, which may be closed if necessary with a plug of clay. The whole arrangement is contained within the niche (Capelle) of the furnace, two sets of condensers in a niche, no part projecting beyond the front wall. The zinc vapor condenses to liquid and collects in the horizontal arm of the pipe, whence it is tapped into ladles by removing the plate which closes the outer end of the pipe. This opening also gives access to the muffle. The uncondensed vapor, together with the carbon monoxide gas, passes up through the short vertical arm of the condenser, and through the incandescent bed of coke burning above it. The object of this is to keep free oxygen from entering the condenser, but that object

[1] German patents, Nos. 8121, 12,821, 28,596 and supplementary patent No. 7411.

appears to be attained only when the coke fire receives a very careful attention, which it is practically impossible to give it. It was also thought originally that any oxidized zinc fume rising through the fire would be reduced and would run back into the condenser, but on the contrary it was found that metallic zinc vapor burned to oxide; zinc vapor may be observed to burn very actively above the grate fire. The carbon monoxide burns of course to carbonic acid above the grate. Since the bed of burning fuel is easily choked by the deposition of zinc oxide, it requires to be poked occasionally and periodically to be renewed. The gases which have passed through the grate rise through a flue in the top of each niche to a common pipe above the furnace, whence they are led to a chimney. The arrangement is shown in the accompanying engraving. All danger of explosions of CO gas is eliminated and the escaping zinc oxide is capable of collection.

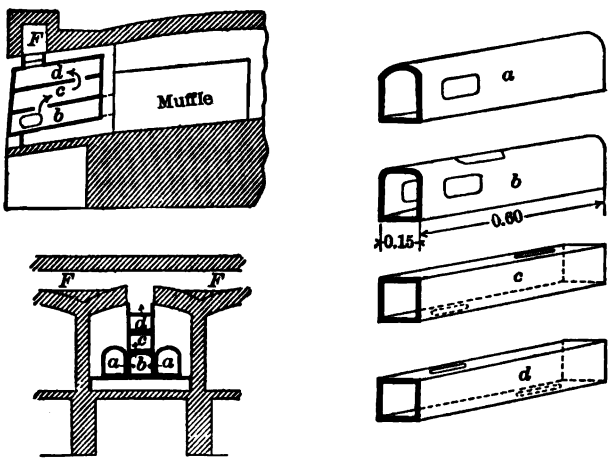
The Kleeman condenser was first introduced in 1879 and was for a time generally used, but was subsequently replaced in most of the works by the Dagner type. The Kleeman condenser is still used, however, at some works, particularly at those of the Schlesische Gesellschaft für Bergbau-und Zinkhüttenbetrieb at Lipine. At those works the condensers fitted with Kleeman grates are provided with small holes in their ends and during the distillation these holes are lightly smeared with clay so that the escaping vapors have to pass through the grate, but if the bed of fuel on the latter becomes filled with a deposit of oxide, so that the vapor cannot find free egress, the increased tension in the condenser blows out the clay stopper at the end and the carbon monoxide gas taking fire the flame gives an unmistakable indication to the smelters that the grate needs attention, either poking the coal or renewing it. The great care which is required with the Kleeman condensers has led, however, to the substitution for them in many works of the simpler Dagner condenser.

Dagner Condenser.—The Dagner condenser is an arrangement which compels the gas from the retort to take a comparatively labyrinthine course before it can escape. The two muffles in a niche are commonly served by a single condensing system. Resting in the end of each retort is a small muffle-shaped pipe, about 1 m. long. Between these pipes is placed a rectangular pipe, making a tight joint with the two main pipes, and having corresponding openings which permit the gas to pass from the main pipes into the middle pipe. Superimposed over the middle pipe are two other pipes of similar form, with corresponding openings, which compel the gas to travel in opposite directions in each member.[1] The general arrange-

[1] In the earlier forms each retort in a niche had its own system, which consisted of three superimposed pipes. The space between the ends was sometimes plastered up with clay in order to make a small combustion chamber within the niche itself.

ment and the form of the various members is shown in the accompanying engraving. The section *b* is 0·15 m. wide and 0·20 m. high; section *c* is 0·12 m. wide and 0·10 m. high; section *d* is 0·10 m. wide and 0·10 m. high, The outer ends of these pipes are closed by iron plates or tiles, which are luted tightly with clay, each plate having a small round hole in the center, sealed with clay. These holes permit the entrance of iron rods in order to dislodge any accumulations of zinc oxide. The openings from one member of the condenser into the next are about 9×10 or 9×12 cm.

The zinc which condenses in liquid form is recovered from the lowest row of pipes, the upper pipes giving only zinc dust and zinc oxide. The gas which finally escapes from the uppermost pipe burns at the opening and

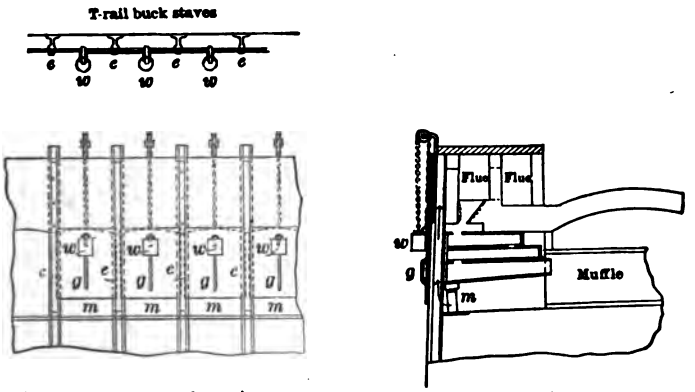


FIGS. 360 TO 365. DAGNER CONDENSER.

the products of combustion, zinc oxide and carbonic acid, are led away through a flue in the top of the niche into a collecting main, whence they are conducted through a dust chamber to the chimney. The front of each niche is closed by a sheet iron plate which may be raised or lowered, permitting the temperature of the condensers to be controlled. There is a small hole in the lower part of the plate which admits the air necessary for combustion of the gas when the plate is lowered. The gas collecting main runs longitudinally over the furnaces, leading to the dust condensing chamber. At certain works the condensing system is developed to the extent of seven chambers with a total length of 110 m., the chambers being provided with vertical walls to lengthen the course that the gas has to travel and increase the depositing surface. In the last chamber there is a

water spray, which precipitates solid particles which are still in suspension, the water being thence led through a coke filter. According to Schnabel[1] this arrangement has proved to be good.

According to Kosmann,[2] the dust deposited in the condensation chambers connected with a Dagner condenser system at the Wilhelminehütte has approximately the following composition: 88·20% ZnO (70·82% Zn), 1·46% CdO (1·27% Cd), 4·44% PbO (4·12% Pb), 4·12% SO_3, 0·05% Mn_3O_4, 1·50% ferric oxide and residue; total, 99·77%. This product is



FIGS. 366 TO 368. ARRANGEMENT OF DAGNER CONDENSERS.

Fig. 366: Plan. **Fig. 367**: Front elevation. **Fig. 368**: Section through condensers.

mixed with fresh ore and recharged into the muffles. The dust collected from the parts of the flues near the furnaces is richer in cadmium and is collected separately for recovery of that metal. At the Wilhelminehütte such dust averages 3 to 4% Cd and sometimes attains 5%. Analyses of flue dust from the Paulshütte are reported by Steger as follows:[3]

	I %	II %	III %	IV %	V %
Zinc oxide	65·71	78·15	83·95	89·92	90·45
Cadmium oxide	7·11	2·09	1·68	3·21	2·76
Lead oxide	3·70	4·29	3·96	0·86	1·38
Ferric oxide	0·50	3·87	0·80	2·10	1·22
Sulphuric anhydride	....	....	....	0·30	0·18
Carbon dioxide	....	....	....	2·44	3·20
Sand	1·98	3·45	1·16	....	....
Loss in ignition	20·42	6·56	8·71	....	....
Insoluble residue	....	....	....	0·55	0·42
Totals	99·42	98·41	100·26	99·38	99·61

[1] Handbuch der Metallhüttenkunde, II, 148.
[2] Zts. f. d. Berg- Hutten-u Salinenwesen im Preuss. Staat, XXXI, 236.
[3] Sammlung chemischer und chemisch-technischer Vorträge, 1896, I, II, 81.

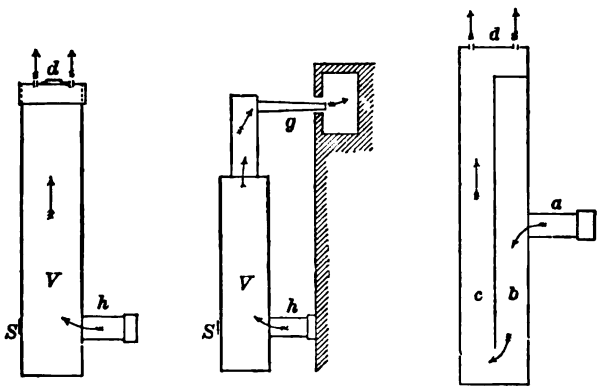
Results from Kleeman and Dagner Condensers.—With the general use of the Kleemann and Dagner condensers in connection with the Silesian zinc furnaces, whereby all the vapor escaping from the clay condensers attached to the retorts is collected in a general condensing system, the inquiry as to what additional recovery of metal is effected naturally presents itself. With respect to that point, however, I have neither been able to find any definite information in the metallurgical literature nor have I been able to obtain it in response to my own inquiries. That there should be an additional recovery of metal would be naturally expected. When the installations were first put in operation in Upper Silesia, however, the metallurgists there found that contrary to their expectations there was no increased recovery, but instead thereof there was an increased loss of metal. This was due to two causes: (1) In operating the main condensing system there was a draught required of course, which draught produced a minus pressure in the retorts, permitting gases of combustion to filter through them and oxidize zinc; while (2) the arrangements for condensation of the fume were imperfect. It is only recently that metallurgists have learned how difficult it is to settle metallic fume and how inefficient even wet methods of condensation may be. In a report upon the practice at the Wilhelminehütte, in 1883, it was stated that not only was it impossible to settle the zinc oxide completely in the largest dust chambers, but also during the emptying of the chambers considerable quantities of dust were lost up the chimney. The difficulty as to the pressure of gas in the furnaces was obviated by providing each furnace with a damper and regulating the draught so that the gas would just be removed as fast as developed, no more. With this method of operation and wet condensation of the fume there has been an increased recovery of zinc, but how much and at what cost it has not been stated publicly.

In the volume on metallurgy of Dammer's *Handbuch der Chemischen Technologie*, p. 673, it is mentioned that the Wilhelminehütte got an additional 0·5% from the ore in the form of oxide of the composition reported by Kosmann. This would imply that 1000 kg. of ore, assaying for example 20% Zn, and therefore containing 200 kg. of zinc, which might yield ordinarily 140 kg. (70%) would with the Dagner condensers give an additional yield of oxide of 5 kg. containing about 3·5 kg. of zinc, corresponding to an increased extraction of 1·75%. Undoubtedly, however, the statement in Dammer's work refers to the practice of a long time ago.

Iron Prolongs.—The prolongs which are used in Belgium and Rhenish Prussia, and at those works in the United States where they have been employed, are generally simple cones or canisters of sheet iron. In Upper

Silesia, however, a great variety of more complicated forms has been employed from time to time, and some of them are still in use by works which make a practice of preparing zinc dust for the market. Although some of these forms of ballons effect apparently a nearly complete condensation of the vapor escaping from the condensers proper, and yield a product which is marketable, they have not displaced the Kleeman and Dagner systems of condensation, notwithstanding the great first cost of the latter and their production of nothing but a between-product, which must be resmelted. The reason for this is no doubt the cumbersome and complicated character of the improved forms, which would entail too much additional care in the management of the furnace.

In designing iron prolongs the chief requisites are the provision of an



FIGS. 369 TO 371. VERTICAL PROLONGS.

ample cooling surface, the insurance that the gas will come in contact therewith as much as possible and the prevention that the fine, condensed dust will be carried off by the escaping stream of gas. The functions and conditions of the prolong reproduce therefore on a small scale those of the dust chamber for a reverberatory furnace, but it is important moreover to avoid forms which are liable to explosions of gas. The iron prolongs which are employed by various works are of three types, namely, the vertical, the horizontal and the combined. The vertical type is adaptable only to the Silesian furnace with one row of retorts. With the Rhenish and Belgian furnaces which have several rows of retorts, either the horizontal type or the combination type may be used, but generally only the former is employed.

Vertical Prolongs.—The simplest form of the vertical prolong is shown

in Fig. 369. It consists of a cylinder *V*, which is connected with the nose of the condenser by the tube *h*. The gas escapes into the furnace house through holes in the cover *d*. In order to avoid such pollution of the at-

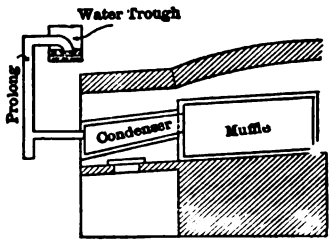


FIG. 372. PALM'S BALLON.

mosphere, the gas is sometimes led by a tube *g*, Fig. 370, into a collecting flue, an arrangement which is preferable to the other. In order to keep the condenser clear, there is an opening *S* in the vertical cylinder, through which a

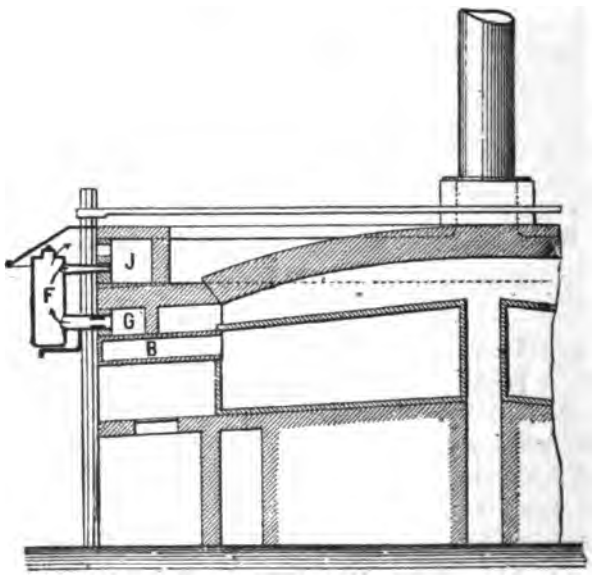
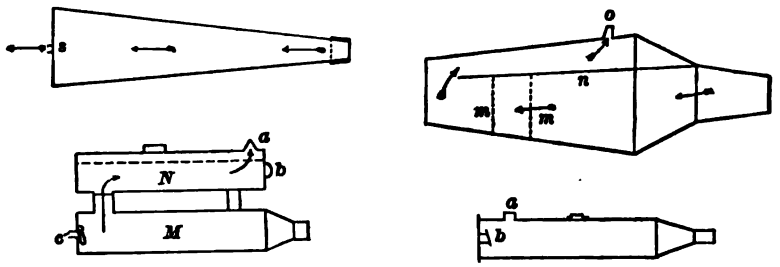


FIG. 373. HAWEL'S BALLON.

rod may be inserted, the opening being closed at other times by a valve or a plug of clay. Another proposed form of vertical ballon has a condensation cylinder and a pipe leading from the latter into the combustion chamber

of the furnace. One would expect that this arrangement would interfere with the discharge of vapor from the retort and condenser, either by producing too strong a suction with direct firing, or too great a back pressure if there be a plenum in the combustion chamber. The cylinder has two openings closed with slides for the removal of the dust collected. Another vertical prolong invented by Palm,[1] is shown in Fig. 372. In this the gas is discharged in a box partly filled with water, the gas being drawn through the water into the combustion chamber of the furnace. The prolong invented by Hawel[2] is shown in Fig. 373. The gas escaping from the condenser enters a chamber *G*, whence it goes through the lower horizontal arm of the ballon, *J* being a flue for the final escape of the gas. In a later form of the Hawel condenser, a tube is interposed between the condenser and the chamber *G*, with better results, it is said. Another type of vertical con-



FIGS. 374 TO 377. HORIZONTAL PROLONGS.

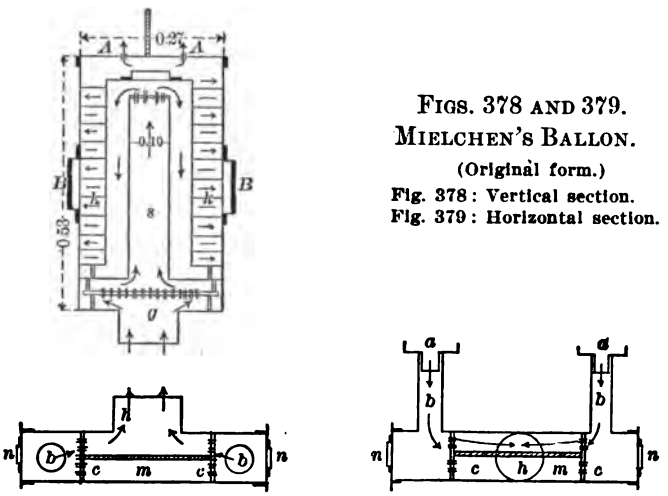
denser is a simple cylinder divided by a diaphragm, as shown in Fig. 371, which compels the gas to take a reversed course and increases its line of travel.

Vertical prolongs are especially liable to gas explosions, the draught in them sucking in air through any leaks there may be and also through the tap hole when the latter is opened for cleaning the condenser. They are free from the danger of explosion only when their temperature is kept below the ignition point of such mixtures of gas and air as may be formed in them. The horizontal prolongs are much less liable to explosion. Many forms of horizontal prolongs have the objection, however, that they discharge the poisonous carbon monoxide gas farther into the furnace house. In order to obviate that, at many works the gas is ignited at the end of the condenser. Sometimes air is admitted at the junction between the condenser and the prolong, in which case the gas escapes from the end of the

[1] German patent, No. 9672.
[2] German patents, Nos. 57,385 and 61,740.

latter partly burned; this method has the disadvantage that the prolongs are run very hot.

Horizontal Ballons.—The simplest forms of horizontal prolongs are a long sheet iron cone of which the large end is slipped over the nose of the condenser and a long canister in the form of a truncated cone, with an escape hole at *s*, as shown in Fig. 374. The latter form is sometimes luted tightly on the nose of the condenser, and sometimes merely set loosely thereon. In the former case the gases must necessarily escape entirely through the hole *s*; in the latter case they escape partly through *s* and and partly at the union. In the case of the large Rhenish and Silesian condensers, connection is made between the latter and the prolong by means



FIGS. 378 AND 379.
MIELCHEN'S BALLON.
(Original form.)
Fig. 378: Vertical section.
Fig. 379: Horizontal section.

of a short tube. Another form of horizontal prolong is shown in Fig. 375. This has a diaphragm *n*, which compels the gas to take a longer course, while in the lower part are placed vertical sieves *mm*, which increase the friction. If such sieves be introduced, a tap-hole at the end of the ballon is of course useless and the ballon is necessarily removed in order to remove obstructions in the condenser. By discharging the gas from the hole *o*, where it is frequently ignited, there is the advantage that it is set free nearer the façade of the furnace and has a better opportunity to escape through the ventilator in the roof of the building. With ballons of this design explosions are rare, though they are not entirely free from danger. A ballon invented by Recha[1] is shown in Fig. 376. This consists of two

[1] German patent, No. 12,768.

superimposed cylinders, of which the lower has a self-acting valve *c*, and the upper a cleaning and explosion cover *b*. The gas escapes through the hole *a* under a hood attached to the furnace. Bugdoll's ballon, Fig. 377, is simply a horizontal cylinder with a valve *b*, and a hole *a* stopped with a filter of some suitable fine material through which the gas escapes.

Combination Ballons.—Of the combination ballons the most important are those invented by Mielchen and Steger. The former, which is used at the Hugohütte, Upper Silesia, is a very elaborate construction.[1] It is illustrated in Figs. 378 and 379. One ballon is provided for each niche of the furnace. The gas from two condensers *aa* passes through two tubes *bb* into the horizontal cylinder *m*, which cylinder has two perforated plates *cc* through which the gas has to pass. The gas then rises through the vertical

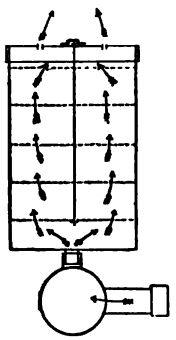


FIG. 380. MIELCHEN'S BALLON.
Modified form.

pipe *h* into a cylinder *s*, which stands on the perforated plate *g*. The top of the cylinder *s* is also closed with a perforated plate through which the gas has to pass, being led thence downward in the direction of the arrows. Fitting closely inside of the external cylinder is a cylindrical coil of sheet iron *kk*, through which the gas rises in the direction of the arrows, escaping finally through the holes *AA*. The external cylinder has hand-holes, *BB*.[2]

The Mielchen ballon described above has been criticized by Doctor Victor Steger in his paper entitled "Verdichtung der Metalldämpfe in Zinkhütten."[3] His criticisms are obviously sound. A slight leakage gives rise to the danger of explosion, which may be serious, while any indentation of the outside cylinder, which may easily happen in handling, binds the cylindrical coil so that it cannot be easily removed. The apparatus is very

[1] German patent, No. 18,635.
[2] Saeger, Hygiene der Huettenarbeiter, Jena, 1895.
[3] Sammlung chemischer und chemisch-technischer Vorträge, I, 11, 62.

heavy, which makes it difficult to handle, and after each period of distillation it must be taken apart and put together again, and the dust arising during its removal is injurious to the men. With careless handling there is no end to the repairs.

A modified form of the Mielchen ballon is shown in Fig. 380. In this the complicated internal arrangement is omitted and the outside cylinder contains simply a series of perforated plates, supported by a rod hanging from the top. These plates present a comparatively large friction surface for

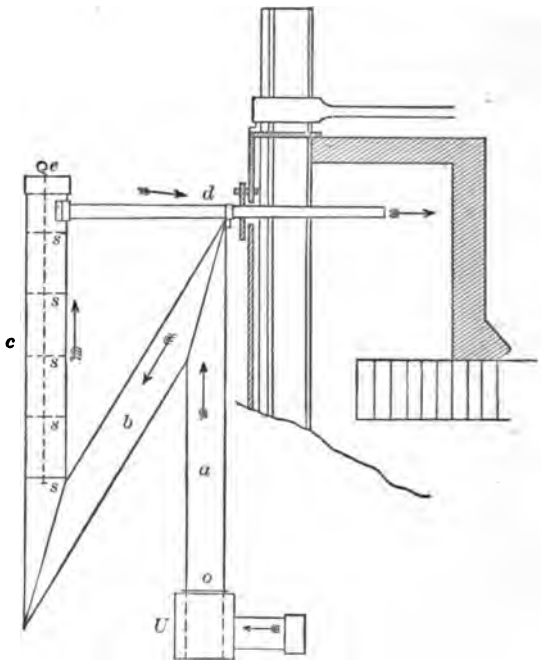
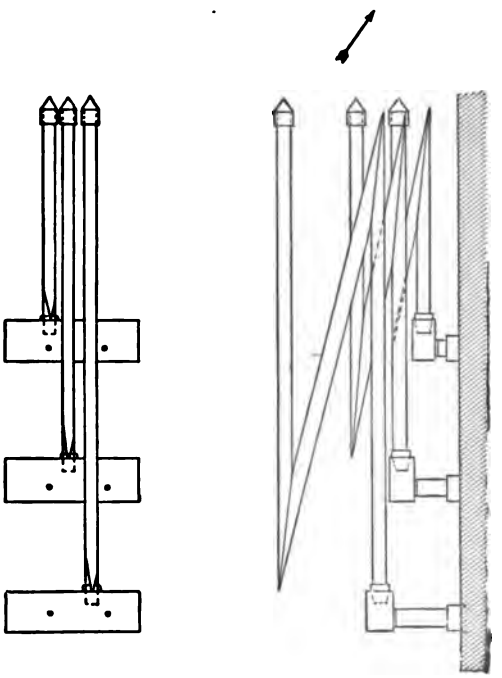


FIG. 381. STEGER'S PROLONG.
Adapted to Silesian furnace.

the deposition of particles of dust and to a considerable extent compel the gas to rise at the periphery of the cylinder where the cooling action of the air is strongest.

The elaborate prolong invented by Steger is in use at the Lazyhütte, near Beuthen, Upper Silesia. It is illustrated in the accompanying engravings which are so obvious that they require no particular description. Fig. 381 illustrates the arrangement with an ordinary Silesian furnace with one row of retorts, the condensers of two retorts being united in a single system.

With the Rhenish furnaces of three rows of retorts, the Steger prolongs are arranged as shown in Fig. 382. The system is also adaptable to Belgian furnaces with their numerous rows of retorts in a similar manner. In any of these arrangements it is necessary to set the vertical members of the ballons so that they will not interfere with direct entrance into the condensers. Steger claims[1] that this is the most efficient of all prolongs, compelling the gas to take the longest and most circuitous course and exposing



FIGS. 382 AND 383. STEGER'S PROLONG.
Fig. 382: Front elevation. Fig. 383: Side elevation.

it to the greatest cooling surface. He claims that it cools the gas from 500° C. to 40° or 60° C. The pipe *c* is provided with perforated plates *ss*, which are hung from the rod *e*. The gas may be burned at *e*, or it may be led through a tube *d* into a flue over the furnace and burned there. It burns with a pure violet flame, indicating that no zinc is present. Owing to the strong cooling of the gas there is no danger of explosion in the condenser.

In adapting his condenser to the Rhenish and Belgian furnaces, Steger

[1] Loc. cit.

makes the tubes which serve the lower rows the longer, if the furnaces are fired in such a way that the lower rows are the hotter; if the upper rows are the hotter, the upper ballons are made the longer. For use with Belgian furnaces, he recommends that five retorts of a row be combined in one system. When it is necessary to tap zinc from the condensers, the entire ballon system must be removed, of course. It is recommended that all the pipes have the same height, as shown in Figs. 382 and 383, in order to insure an equal removal of the gas. However, this prolong has not yet found application with Belgian furnaces.

XIII.

REFINING IMPURE ZINC AND COMPOSITION OF COMMERCIAL SPELTER.

In general the crude spelter drawn directly from the condensers is sufficiently pure to be utilized without refining. This is not due, however, to chance, but on the contrary to a careful selection of the ores to be distilled so that the percentage of impurities, especially lead, which may contaminate the spelter is limited. This is possible only when sufficient good ore is available to dilute the impurities of the inferior ore to the proper degree; when only one kind of ore is to be had the character of the spelter will correspond to the character of the ore. If the ore contain only a moderately high percentage of lead the spelter will also contain lead and frequently too much to make the metal marketable without a previous refining.[1] In speaking of refining impure zinc, the removal of excessive lead and iron contents alone are referred to, since those are not only the commonest and most objectionable impurities, but also they are the only ones of which the separation can be readily and economically effected. That is done by remelting the crude spelter in a reverberatory furnace, whereby the larger part of the lead, by reason of its greater specific gravity (11·4) separates from the lighter zinc (sp. gr. 6·86 to 7·16) and settles to the bottom, while an iron alloy (hard zinc) separates between the lead and the zinc, and mechanical admixtures and oxides collect on the surface as dross. The extent to which the iron content of ordinary spelter can be reduced by gravity separation appears to be, however, rather small.

The foregoing remarks as to the purity of the spelter being determined by the character of the ore require some qualification, inasmuch as the spelter made from the same charge of ore will be found to vary in composition according to the conditions of the distillation and the methods of handling the molten metal. In general the spelter distilled at a comparatively low temperature is purer than that which is distilled at a high temperature. For that reason the first drawing of metal is likely to be of

[1] Refer also to page 503.

better quality than the second and third, and it is sometimes worth while to keep it separate, as is done by certain smelters at Iola, Kan. The method of handling the molten metal is also important. All ordinary spelter receives undoubtedly a part of its iron content from the tools by which it is drawn from the condensers and the kettle in which it is collected, the ability of molten zinc to form an alloy with iron being demonstrated in the process of galvanizing. At Pulaski, Va., where it is aimed to produce a spelter absolutely pure from iron, or almost so, the molten metal is never permitted to come in contact with an iron tool or vessel.

The value of spelter increases in proportion to its purity. In order to possess the high degree of fluidity which is indispensable for the casting of perfect statuary or works of art, and the ductility and tenacity necessary for such fine sheet brass as is used in making cartridges and for spinning brass, the spelter must be practically free from all other metals. High grade spelter is also required for photo-engraving plates, for the purpose of coining and as a chemical reagent.

The Société Anonyme de la Vieille Montagne classifies its spelter as follows: (1) Zinc extra-pur; (2) zinc fonte d'art; (3) zinc laiton; (4) zinc à galvanisation. The first is almost chemically pure and corresponds to the American high grade spelters. The second is a superior grade of metal, used for casting objects of art. The third and fourth are respectively the ordinary grades of metal supplied for brass-making and for galvanizing iron.

Miscibility of Lead and Zinc.—The miscibility of molten lead and zinc and the saturation point of the solutions conform to certain physical laws, which were discussed by E. Heyn in *Berg- u. Hüttenm. Ztg.*, LIX, 47, 559 (November 23, 1900); reference should be made to that paper, which is chiefly of scientific interest, by those who desire to investigate this subject more minutely. Spring and Romanoff determined the solubility of lead and zinc at various temperatures as shown in the following table:

Temperature °C.	Lower Stratum		Upper Stratum	
	% Pb.	% Zn.	% Pb.	% Zn.
334	98·8	1·2	....	....
419	....	....	1·5	98·5
450	92·0	8·0	....	....
475	91·0	9·0	2·0	98·0
514	89·0	11·0	3·0	97·0
584	86·0	14·0	5·0	95·0
650	83·0	17·0	7·0	93·0
740	79·0	21·0	10·0	90·0
800	75·0	25·0	14·0	86·0
900	59·0	41·0	25·5	74·5

This table shows how the percentage of lead alloyed with zinc increases with the temperature. At a point 2° or 3° above the melting point of zinc the latter still retains 1·5% Pb. This result is confirmed by the experiments of Roessler and Edelmann. In refining lead-bearing zinc by gravity on a large scale it is found, however, that the lead content can be reduced to the limit of about 1%; further than that it does not appear possible to go. For example, a crude Silesian spelter assaying 2% Pb was refined down to 1·125% Pb; a portion was then kept for some hours in a molten condition in a crucible 45 cm. deep, after which samples taken from various levels above the bottom of the crucible were assayed, with the following results:[1]

0 cm.	8 cm.	14 cm.	22 cm.	31 cm.	39 cm.	45 cm.
1·120%	1·124%	1·125%	1·119%	1·121%	1·123%	1·125%

How difficult it is to effect a further separation of lead from zinc is shown by an experiment by Ross.[2] He permitted a pipe 1·3 m. high, filled with molten zinc at red heat, to cool gradually during eight days, when the upper part of the bar still contained 0·5% Pb; experiments in centrifugal separation gave no better results. Kraut[3] asserts that the percentage of lead in the plates from the same pouring from the same furnace varies, and remains irregularly distributed even after the remelting, so that a correct test sample (a ladle sample is best) is hardly possible, and the purchaser, in order to protect himself, must stipulate that in the remelting of the zinc no portion rich in lead shall settle to the bottom.

In refining zinc by gravity separation, the temperature of the bath not only influences the chemical composition of the zinc, but also its physical character. For instance, satinized sheets, which must possess luster and hardness, for the smoothing of paper are manufactured from metal heated higher than usual, and left to settle thoroughly, and the sheets are rolled oftener and more slowly at a temperature lower than usual; tanners' sheets, which serve as an underlay in the working of superior skins, are made of a specially soft and gradually cooled zinc.

Practice in Refining.—The refining of spelter is something which is unpracticed and unknown in the United States, where the ore supply is generally of a superior quality and even the lowest grades of spelter produced are comparatively low in lead. In Upper Silesia on the other hand nearly all of the crude spelter is comparatively high in lead and con-

[1] Steger, Sammlung chemischer und chemisch-technischer Vorträge, 1896, I, ii, 79.

[2] Berg-u. Hüttenm. Ztg., 1894, p. 320.

[3] Zeits. f. analyt. Chem., 1886, p. 269.

sequently requires to be refined. It is natural, therefore, that the refining process has received its highest development at those works. The methods followed at the sheet zinc rolling mills at Birmingham, England, are, however, quite similar to the Silesian.

The crude Silesian spelter always contains lead, the amount being generally from 1·25 to 3%. The crude spelter of the Wilhelminehütte carries about 3% Pb, and that of the Hohenlohehütte the same amount. The Guidottohütte at Chropaczow, on the other hand, produces a comparatively pure metal, which contains only 1·3% Pb. Analyses of various crude Silesian spelters are given in the following table:

	I %	II %	III %	IV %	V %	VI %	VII %	VIII %	IX %	X %	XI %
Zn.	99·588	98·452	98·441	98·182	97·243	98·143	97·530	97·3340	98·3631	98·123	98·296
Pb.	1·326	1·493	1·470	1·705	2·171	1·742	2·409	2·6140	1·5240	1·853	1·672
Fe.	0·056	0·033	0·033	0·057	0·054	0·039	0·031	0·0300	0·0995	0·020	0·023
Cd.	0·030	0·018	0·056	0·056	0·532	0·076	0·030	0·0206	0·0110	0·003	0·004
As.	tr.	0·002	tr.	tr.	tr.	.....	.....	0·0014	0·0024	.....	.....
Ag.	.....	0·002	.....	.....	.....	.....	.....	.....	.....	.....	.....
C.	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.001	0·005
Totals.	100·000	100·000	100·000	100·000	100·000	100·000	100·000	100·0000	100·0000	100·000	100·000

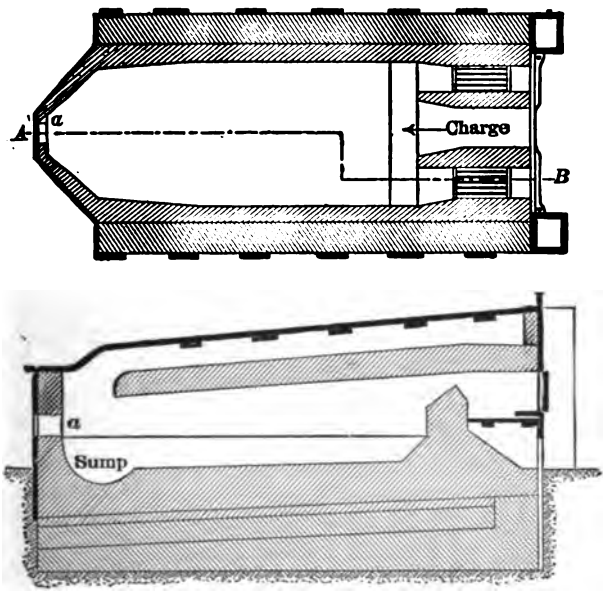
I. Spelter from roasted blende, Paulshütte.
II. Spelter from calamine, Paulshütte.
III and IV. Spelter distilled from a charge of 25% roasted blende and 75% calamine, Paulshütte.
V, VI and VII. Spelter distilled from oxidized flue dust, Paulshütte.
VIII and IX. Spelter from another works.
X and XI. Spelter from Rosamundehütte.

Remelting Furnaces.—For the remelting of the crude zinc there are employed in Upper Silesia reverberatory furnaces 5 to 6 m. long, 2·75 to 3·2 m. broad in front, 2·75 to 3·2 m. at the back, and 1·8 to 1·9 m. high,[1] with two grates at one of the ends and a hearth of tamped clay sloping about 3° toward the ladle door at the other end, in front of which a sump is hollowed in the hearth. Between the two fireplaces is the charge-opening, from which a sloping, longitudinally corrugated iron plate leads to the hearth. The sump in the other end of the hearth is sometimes separated from the furnace chamber by a thick slab of refractory material, the bottom of which resting upon fire brick in the hearth dips into the bath of metal, whereby the impurities floating on the surface of the metal are kept out of the ladling pot. The gases of combustion pass from the fireplaces over the hearth and through flues in the arch into a canal over the latter, and thence to a chimney 20 to 25 m. high. Frequently the products of com-

[1] Berg-u. Hüttenm. Ztg., 1873, 290; ibid., 1877, p. 100.

bustion are caused to circulate around chambers, in which the zinc to be rolled is heated, before reaching the chimney.

The construction of such a furnace is shown in Figs. 384 and 385.[1] The hearth is formed of lean clay rammed down upon a brick base. The door through which the refined zinc is ladled out is shown at *a*. Another form of furnace, with only one grate, is shown in Fig. 386, in which *a* is the grate, *b* is the bed of fire clay, *c* is the sump, *e* is the door through which the metal is ladled out, *iii* are chambers heated by the flame, in which zinc plates and sheets can be heated; and *f* is the flue leading to the chimney.



FIGS. 384 AND 385. SPELTER REFINING FURNACE, WITH TWO GRATES.

The crude zinc is charged into the furnace through a door in one of the long sides, near the fire bridge. The hearth is 4·7 m. (15 ft. 5 in.) long and 2 m. (6 ft. 6 in.) wide; the lowest point of the hearth is 0·5 m. (1 ft. 8 in.) below the door out of which the metal is ladled. Such a furnace usually contains from 28 to 30 metric tons of zinc, and is capable of refining 9 to 10 tons in 24 hours, with a consumption of 900 kg. of coal. A furnace used at the Hohenlohehütte is shown in Figs. 387 to 389. Its length is 6 m. (19 ft. 8 in.) and its width 3·2 m. (10 ft. 6 in.); there are two grates and between them is the charging door. The products of com-

[1] Berg-u. Hüttenm. Ztg., 1873, p. 290.

bustion traverse the furnace lengthways from the grate and escape through openings in the roof as shown. This furnace holds 30 metric tons of zinc.

Method of Operation.—The slabs of crude zinc are preheated upon the corrugated iron plate, whence they are gradually pushed into the bath of molten metal. A full charge is from 20 to 30 metric tons of zinc, which is fed in during two or three days; the temperature is not raised much beyond the fusing point of the zinc. About every 12 hours the dross, consisting chiefly of zinc oxide, which floats on the surface of the bath, is raked upon the sloping iron plate at the charge hole, where it is mixed with sal-ammoniac, in the proportion of about 0·25 kg. per 100 kg., to free it from

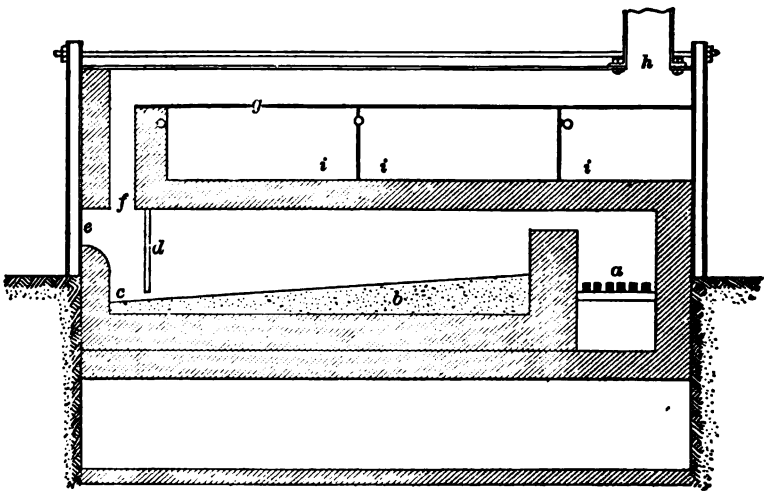
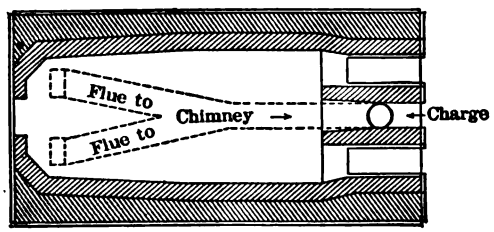


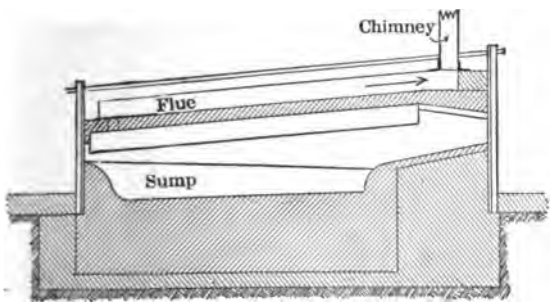
FIG. 386. SPELTER REFINING FURNACE, WITH ONE GRATE.

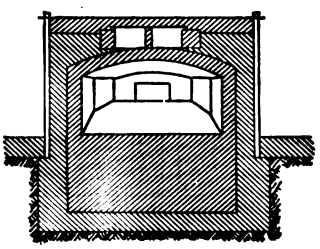
adhering particles of zinc, after which the residue is withdrawn through the charge hole.

In order to avoid oxidation of the zinc, which is of course highly undesirable, the crude spelter must be melted very gradually, at the lowest possible temperature and the flame kept as reducing as possible, i.e., the coal is burned with a deficiency of air so that the products of combustion will be rich in carbon monoxide. When the furnace has received its full charge of 20 to 30 tons of zinc, which requires two or three days in starting up a new campaign, the ladling out of the metal is commenced, as much fresh zinc being then charged in as is ladled out, or approximately 9 to 10 tons per 24 hours. From the bath of molten metal, which is kept at a constant

level, the lead settles to the bottom, while the iron forms with the zinc and a portion of the lead a difficultly fusible alloy which floats above the lead, and uppermost is the stratum of pure zinc. By means of an iron rod, the pure zinc, the hard zinc, and the lead are easily distinguishable from each other, the metallic bath feeling soft as far as the molten zinc extends, while the layer of hard zinc feels mushy, somewhat like ice at its melting point. The layer of molten lead feels soft again. The mechanical impurities, which have a lower specific gravity than the zinc, separate out and float upon the surface of the bath together with such zinc as is oxidized.







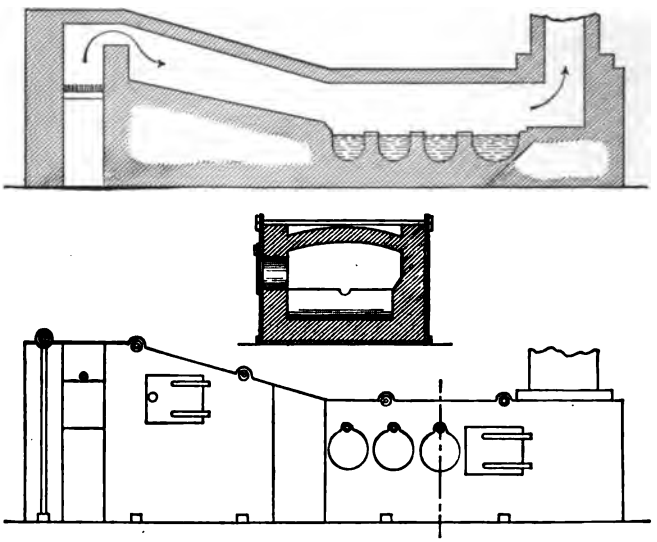
FIGS. 387 TO 389.
REFINING FURNACE, USED AT HOHENLOHEHÜTTE.
Fig. 387: Horisontal section.
Fig. 388: Longitudinal vertical section.
Fig. 389: Transverse vertical section.

These skimmings are removed from time to time. When properly worked, their quantity does not exceed 1·5% of the zinc charged.

For emptying the furnace it was the custom formerly, after the dross had been raked away, to ladle out first the zinc and then the lead, after which the furnace was recharged with a fresh lot of crude spelter. This practice, however, caused a great expenditure of fuel and loss of time, and was abandoned in favor of the present method, which is to draw off the lead underlying the zinc about once a week, without interrupting the work, and to recharge continuously with fresh zinc. In some cases the lead is removed by an endless screw[1] working in a cast iron pipe, 1·3 to 1·4 m.

[1] Berg- u. Hüttenm. Ztg., 1883; 1888, p. 422; 1890, p. 130.

long and 0·12 m. diameter, which is inserted through one of the doors on the long side of the furnace. The screw has about 12 turns and 90 to 100 mm. pitch. The pipe has a spout at its upper end through which the lead runs off. The screw is turned by means of a crank. This apparatus is employed at Lipine. Another device[1] is to insert in the bath of molten metal a large iron or clay pipe, closed at the lower end with a clay plug; when the latter is knocked out the lead rises in the pipe, whence it is ladled out. At the Hohenlohehütte 5000 kg. of zinc are ladled out every 12 hours, and the same quantity of crude zinc is added; the lead settled out is removed every eight days.



FIGS. 390 TO 392. ASH'S SPELTER REFINING FURNACE.

Fig. 390: Longitudinal section through center. Fig. 391: Transverse section on the dotted line of Fig. 392. Fig. 392: Side elevation.

When the zinc is to be drawn, it is left to stand for about one hour, so that it may assume a uniform temperature; it is then poured with galvanized cast iron ladles into cast iron molds. It saves time and labor to have the molds so that they can be brought under the spout one after another standing upon a revolving frame, or else upon a car moved by a suitable mechanical device as in lead refineries. Another way is to arrange the molds in a semi-circle and fill them from a movable spout. The zinc is skimmed in the molds while still fluid, and then is covered and left to cool gradually, whereby the metal becomes more fibrous and pliant. When

[1] Berg- u. Hüttenm. Ztg., 1890, p. 130.

the refining is done in connection with a rolling mill the plates of refined metal are sent to the breaking-down rolls while still hot.

The removal of the layer of hard zinc which accumulates in the refining furnace is generally done during an idle time of the furnace, for instance, when repairing. All of the zinc is first ladled out, the lead is then pumped out, and finally the pasty mass of ferruginous metal is removed with perforated ladles, so that the entrained lead may drip off.

The consumption of coal in zinc refining ranges from 7·5 to 10% of the weight of the metal melted. The percentage of marketable lead recovered depends of course upon the tenor of lead in the crude spelter. At the Silesia rolling mill in 1900 there were obtained 330 tons of lead from 17,843 tons of spelter; a yield of 1·85% The zinc ashes and other between-products amounted to 208 tons, i.e., about 1·16% of the weight of the crude spelter. The ordinary furnace, of 9000 to 10,000 kg. capacity, requires the labor of one smelter and one helper per shift of 12 hours.

Quality of Metal Produced.—The refining of crude spelter by gravity separation enables the lead contents to be reduced from 3% or more to about 1 to 1·25%. This is the general result of the practice in Upper Silesia. The spelter ladled from the sump nearest the underlying lead is richer in the latter metal than that which is got from near the top. This is usually cast into slabs and sold separately as second-class metal. The lead drawn from the furnace still contains a percentage of zinc and is sold for purposes for which that impurity presents no objection; or else the lead is refined by burning off the zinc. A specimen of such lead assayed 4·24% Zn, 0·0008% Fe and 95·76% Pb. The hard zinc, rich in iron, is sold for the manufacture of Delta metal, Sterro metal and similar alloys. The oxide skimmings are mixed with ore and redistilled.

Ash's Furnace.—A. J. Ash has patented [1] the type of furnace shown in Figs. 390 to 392, which needs no special description. The metal passes from one sump to the next, the impurities gradually settling out, giving refined metal in the last sump.

Direct Refining.—The metal drawn from the condensers of the distillation furnace may undergo a partial separation of its lead in the collecting kettle, before being cast into molds, so that the portion which is last poured off will be less pure than that which is first decanted. A considerable quantity of lead is separated from the spelter in that manner in Upper Silesia. This idea has been developed further by Emil Herter, of Beuthen, Upper Silesia, who has lately patented an improved apparatus for separating lead from spelter simultaneously with the drawing of the latter from the con-

[1] United States patent, No. 702,526, June 17, 1902.

densers of the distillation furnace. The molten spelter is tapped into a kettle lined with refractory material, which has a lip on one side and at the bottom an outlet hole fitted with a goose-neck, or any other suitable device, for drawing off the lead settled from the spelter. The kettle is surrounded by a jacket which is arranged with fire-bars at the bottom, and the annular space between the kettle and the jacket is kept filled with glowing coke. A cover is provided to close the annular division at the top. The whole apparatus is supported on trunnions at the end of a horizontal arm which is mounted on a movable carriage. The zinc is allowed to remain in the kettle at any desired temperature for as long a time as may be necessary to permit separation of the lead.[1]

SEPARATION FROM ZINC OF IMPURITIES OTHER THAN LEAD.—Numerous methods have been proposed for separating from zinc impurities other than lead in the dry way, but most of them are inapplicable on a large scale, and such as are in use are only for the preparation of very pure spelter for chemical or other special purposes. Richards[2] adds aluminum to the molten zinc and permits the latter to stand for some time, whereby the impurities other than lead separate and rise to the surface. L'Hote,[3] in order to expel antimony and arsenic, stirs into the molten metal 1 to 5% of anhydrous magnesium chloride to volatilize antimony and arsenic chlorides. A simple redistillation of the crude spelter serves as a refining process, but unless special precautions are taken the loss of zinc by failure to condense makes the process unprofitable. Thus an experiment at an American works showed that a product assaying 99·90% Zn could be obtained from crude Western spelter by redistillation, but the loss amounted to 8 or 9%. Harnickel distils the zinc under strong pressure and condenses the vapor in vacuo, whereby the more volatile constituents go off. Lescolur first smelts the crude spelter with saltpeter and then with zinc chloride, driving off antimony, arsenic, sulphur and phosphorus; but iron, lead and copper cannot be expelled. Selmi, in order to remove the arsenic, dips a piece of sal-ammoniac, fastened to an iron wire, once or twice into the molten zinc, holding it firmly against the bottom; zinc oxide is also reduced thereby. Cumming smelts granulated zinc with a mixture of sulphur and soda.

Stolba mixes three parts calcined gypsum with one part pulverized sulphur and water into a thick paste, forms the mass into balls of about 5 cm. diameter, dries them on a wooden stick, and dips them into the molten zinc, when sulphur and water vapors escape with a violent commotion of

[1] British patent, No. 8175, Apr. 20, 1901.
[2] Comptes Rendus, XCVIII, 1491.
[3] United States patent, No. 448,802.

the metal bath; after a while the outer crust of the balls is taken off and the operation repeated several times, after which zinc free from arsenic remains, though only traces of iron and lead are removed. Merton for removing copper smelts zinc together with sulphur, skims off the metallic sulphides which are formed and then distils the zinc under pressure, pouring it into molds cooled by water. According to Funk, sulphur and carbon may be removed by repeated melting of the zinc and filtering through an asbestos funnel. If molten zinc be stirred with a greased wooden rod and brimstone be added, metallic sulphides will separate and collect on the surface; the addition of the sulphur is continued until a sample shows no arsenic with a Marsh apparatus. Ende thinks, however, that traces of fat may by this process remain in the zinc, which in the reduction of ferric oxide salts with zinc and subsequent titration of ferrous oxide with potassium permanganate may cause disturbance.

Composition of Commercial Spelter.—The composition of the principal commercial spelters is shown in the subjoined tables, from which it will be observed that the range is within comparatively narrow limits. The American high grade spelter contains only 0·01 to 0·02% Pb and 0·01 to 0·02% Fe. The Vieille Montagne high grade metal is about as good. Electrolytic spelters are slightly inferior. American Western spelter ranges from about 0·4% Pb and 0·02% Fe in the better brands up to about 1% Pb and 0·05% Fe in the poorer. Silesian spelter is not quite so good as the ordinary Western.

COMPOSITION OF AMERICAN HIGH GRADE SPELTER.

Reference	Zn %	Pb %	Fe %	Cd %	As %	Sb %	Cu %	Bi %	Ag %	SiO_2 %	S %
I.	99·981	tr.	0.019	.....	.....	.....	.....	.....	.....	.....	tr.
II.	99·953	0·027	0.0200	.....	.....	.....	.....	.....	.....	.....	.....
III.	99·378	.....	0.0410	0·078	.....	.....	0·530	.....	.....	.....	.....
IV.	?	.....	0.0405	.....	.....	.....	.....	.....	.....	0·2390	.....
V.	99·800	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
VI.	99·500	0·410	0·0900	.....	.....	.....	.....	.....	.....	.....	.....
VII.	99·960	0·020	0·0200	.....	.....	.....	.....	.....	.....	.....	.....
VIII.	99·9629	0·0225	0·0121	.....	.....	.....	.....	.....	.....	0·0019	0·0006

I. Bertha Zinc Co's high grade product (E. C. Moxham, Eng. & Min. Journ., Nov. 25, 1893).

II. Passaic Zinc Co., analyst unknown (Schnabel, Handbuch der Metallhüttenkunde, Vol. II).

III, IV. Lehigh Zinc Co., analyst unknown. These are not properly classed as high grade spelter, but are included with this group for convenience. The samples are remarkable for their high tenor in copper and silica, respectively; they are in no way typical of metal made at Bethlehem in recent years.

V, VI. Passaic Zinc Co. (1894).

VII. Lehigh Zinc Co.

VIII. Bertha Zinc Co., analysis of F. P. Dunnington, 1881.

COMPOSITION OF AMERICAN WESTERN SPELTER.

Reference	Pb %	Fe %	Cd %	As %	Sb %	Cu %	Bi %	SiO_2 %	S %	C %
I.	0·0701	0·7173	.....	0·0603	0·0249	0·1123	.....	0·0346	0·0035	0·1775
II.	0·0061	0·2863	.....	0·0590	.....	0·0013	.....	0·1374	0·0741	0·0016
III.	0·6531	0·0095	0·0056	0·0353	.....	.....	.....	.....	tr.	.....
IV.	0·6295	0·0315	tr.	tr.	tr.	.....	.....	.....	tr.	.....
V.	0·6725	0·0546	0·0011	.....	tr.	.....	.....	.....	tr.	.....
VI.	0·4105	0·0523	tr.	tr.	tr.	.....	.....	.....	tr.	.....
VII.	0·8723	0·0233	tr.	.....	.....	.....	.....	.....	0·0601	.....
VIII.	0·3063	0·0283	tr.	.....	.....	.....	.....	.....	tr.	.....
IX.	0·5875	0·0357	tr.	.....	tr.	.....	tr.	.....	tr.	.....
X.	0·3765	0·0390	tr.	.....	tr.	.....	.....	.....	tr.	.....
XI.	0·2513	0·0475	0·0188	tr.	.....	.....	.....	.....	.....	.....
XII.	0·3990	0·0320	0·2120	.....	.....	.....	.....	0·0020	0·0010	.....
XIII.	0·8100	0·0200	tr.	.....	.....	.....	.....	.....	.....	.....
XIV.	0·5460	0·0260	0·0190	.....	.....	.....	.....	.....	.....	.....

I and II, Missouri spelter, old analyses (Pack). III to XI, prime western spelter reported by W. H. Seamon, in Eng. & Min. Journ., Nov. 14, 1896, and by Suppan (Bulletin of the Missouri Mining Club, Vol. I, No. 2, p. 51). Sample III was from the Glendale works of the Edgar Zinc Co., So. St. Louis (Carondelet), Mo.; IV, from Cherokee, Kas.; V, from Nevada, Mo.; VI, from Pittsburg, Kas.; VII to X, from Peru, Ill.; XI, from Lasalle, Ill.; XII, from Wenona, Ill., analyzed by A. W. Palmer; XIII and XIV, spelter produced by Cherokee-Lanyon Spelter Co., Kansas, 1900.

COMPOSITION OF BELGIAN AND RHENISH SPELTERS.

Reference	Zn %	Pb %	Fe %	Cd %	As %	Sb %	SiO_2 %	S %	C
I.	.....	1·460	0·022	.....	.....	.....	.....	.....	.....
II.	99·031	0·708	0·023	0·017	0·006	0·175	0·002	0·022	0·016
III.	99·948	0·040	0·012	nil	nil	nil	.....	.....	.....
IV.	97·890	2·000	0·040	0·070	tr.	tr.	.....	.....	.....

I. From Birkengang works, Stolberg.

II. Reported by W. H. Seamon as a spelter excellently adapted to sheet-rolling; sample presumably from the Soc. Anon. de la Vieille Montagne.

III and IV. Range of Vieille Montagne spelter employed for the manufacture of zinc white (Ad. Firket, Annales des Mines de Belgique, VI, ii, 210).

COMPOSITION OF SILESIAN SPELTER.

Reference	Zn %	Pb %	Fe %	Cd %	As %	Sb %	Cu %	Bi %	Ag %	S %
I.	98·800	1·080	0·023	0·0340	0·008	tr.	.....	.....	.....	.....
II.	98·910	1·030	0·017	0·0150	0·002	tr.	.....	.....	.....	.....
III.	98·497	1·4483	0·028	0·0245	tr.	.....	0·0002	tr.	0·0017	tr.
IV.	98·193	1·7772	0·028	.....	.....	tr.	.....	.....	tr.	0·002
V.	98·783	1·1921	0·0238	.....	.....	tr.	0·0002	tr.	0·0007	tr.
VI.	98·240	1·7000	0·0200	.....	.....	.....	.....	.....	.....	0·040
VII.	98·782	1·1180	0·0240	0·0120	0·042	0·022	.....	.....	.....	.....
VIII.	98·930	1·0000	0·0300	0·0180	0·012	.....	.....	.....	.....	.....
IX.	98·775	1·1180	0·0200	0·0250	tr.	.....	.....	.....	.....	.....
X.	98·852	1·1000	0·0300	0·0180	tr.	.....	.....	.....	.....	.....
XI.	98·833	1·1240	0·0240	0·0172	0·0019	.....	.....	.....	.....	.....
XII.	98·868	1·1100	0·0200	0·0010	.....	.....	.....	.....	.....	.....
XIII.	98·918	1·0620	0·0170	0·0020	.....	.....	.....	.....	.....	.....

I and II. Range in composition of Silesian spelter in August, 1898 (The Mineral Industry, VII, p. 736).

III. From Georgshütte, analyzed by Schneider and Peterson (Wagner's Chemical Technology).

IV. From Georgshütte, marked CH, analyzed by Schneider and Peterson (Wagner's Chemical Technology).

V. From Bergwerksgesellschaft G. von Giesche's Erben (Reckehütte), analyzed by Schneider and Peterson (Wagner's Chemical Technology).

VI. From Hohenlohehütte; evidently crude spelter.

VII. From Lazyhütte, reported by Steger; rated a very good grade of metal.

VIII to X. From Paulshütte, reported by Steger. Sample No. VIII also contained 0·01% thallium.

XI. From Lazyhütte, reported by Steger; rated a very good grade of metal.

XII and XIII. From Beuthenerhütte, according to E. Jensch.

COMPOSITION OF OTHER EUROPEAN SPELTERS.

Reference	Pb %	Fe %	Cd %	Reference	Pb %	Fe %	Cd %
I.	0·633	0·032	0·054	IV.	0·536	0·018	0·069
II.	0·541	0·010	....	V.	1·100	0·150	....
III.	0·3239	0·0253	....	VI.	1·030	0·040	....

I. From Sagorhütte, Austria, analyzed by Schneider and Peterson (Wagner's Chemical Technology).

II. From Sagorhütte, Austria, analyzed by Prcivoznik (Mittheilungen aus dem K. K. General Probiramt in Wien, 1890-91).

III. From Cilli, Styria, Austria.

IV. From Johannisthal, Carniola, Austria.

V. From Dombrowa, Poland.

VI. From Freiberg, Saxony, according to Doctor Föhr. This sample also contained 0·07% Sn.

COMPOSITION OF ELECTROLYTIC SPELTERS.

Reference	Zn %	Pb %	Fe %	Cd %	As %	Sb %	Cu %	Bi %	Ag %
I.	99·93	0·0600	0·01	.....	.....	.....	.....	.....	.....
II.	99·9446	0·0341	0·0099	.....	tr.	tr.	0·0114	.....	tr.
III.	99·2300	0·3500	0·2300	.....	.....	.....	0·0700	.....	.....
IV.	99·93	.....	.....	.....	.....	.....	.....	.....	.....
V.	99·98	.....	.....	.....	.....	.....	.....	.....	.....
VI.	99·98	0·0200	.....	.....	.....	.....	.....	.....	.....

I. Spelter produced in 1893 by the Nahnsen process at the Silesiahütte, No. II, at Lipine, Upper Silesia (private notes).

II. Spelter produced in 1893 in connection with the Roessler-Edelmann process of lead desilverization (reported by the Roessler & Hasslacher Chemical Co., of New York).

III. Spelter produced in 1897 by the Ashcroft process at Cockle Creek, N. S. W.

IV and V. Spelter produced by the Hoepfner process at Winnington, England (1898).

V. Spelter produced at Duisburg, Germany (Oest. Zts., 1895, p. 123.)

ELECTROLYTIC REFINING OF ZINC.—The electrolytic refining of impure zinc, using sheets of zinc as the cathode and plates of the impure zinc to be refined as the anode, is analogous to the electrolytic refining of impure cop-

per, but simple as the method appears from the theoretical standpoint it is attended by numerous practical difficulties which are so serious that the process has not yet proved applicable on a large scale, notwithstanding the unquestionable usefulness which such a process, if capable of economical performance, would have for the refining of the zinc contaminated with lead which is produced in Upper Silesia and elsewhere. The difference between a process for the electrolytic extraction of zinc from ores and the electro-deposition of zinc from an alloy consists in the supply of the zinc in the former case in the electrolyte, insoluble anodes being employed,[1] while in the latter case the supply of zinc in the electrolyte would be derived from a soluble anode. With the employment of soluble anodes the electrical energy developed thereat is equivalent to the energy required for the deposition of zinc at the cathode, wherefore all that has to be supplied is the quantity necessary to overcome the resistance of the electrolyte, effect the migration of the ions and make good the loss by leakage; consequently the requirement of extraneous energy is much less where soluble anodes can be employed than where the use of insoluble anodes is necessary. Notwithstanding the low voltage required with soluble anodes as compared with insoluble anodes, however, the electrolytic refining of impure zinc has so far failed to be economically successful, chiefly because of the difficulties involved in the removal of the impurities and the great care and skilled labor required in the production of a deposit of zinc that can be rolled. The result of the experience is that although impure zinc can be refined electrolytically, up to the present time it has not proved commercially successful to do so, except in the case of the impure zinc crusts obtained from the desilverization of lead, which is a special process of comparatively little importance in the metallurgy of zinc proper.

Nahnsen Process.—In Upper Silesia the electrolytic refining of impure zinc by the Nahnsen process has been tried on a large scale at the Silesiahütte at Lipine. The product handled there has been the spelter ladled from the sump of the ordinary refining furnace nearest to the underlying lead, which is richer in the latter metal than the spelter drawn from the top of the bath; this leady product is usually cast into slabs separately and sold as second-class metal. For the electrolytic refining of it by the Nahnsen process it is cast into slabs to serve as anodes, similar in shape to the slabs of crude copper used for anodes in copper refining. The Nahnsen process as applied at Lipine consists in the electrolysis of such anodes in an electrolyte composed of a mixture of zinc sulphate and potassium sulphate,

[1] The processes for the extraction of zinc from ores by the employment of soluble anodes have given absolutely no promise of success.

or some other alkaline sulphate, maintained at a temperature of 122° to 140° F., the impurities taken into the electrolyte from the anodes being precipitated by means of zinc dust. The electrolyte contains 1·375 to 2·75 oz. (39 g. to 78 g.) of zinc sulphate and 10·75 to 5·375 oz. (304 to 152 g.) of alkaline sulphate per liter. The quantity of zinc dust necessary to precipitate the impurities is calculated from the composition of the crude spelter. Thus if the latter were contaminated with 1% Cu the consumption of zinc dust for the deposition of 1000 kg. of pure zinc would be 11 kg. An electromotive force of 0·5 to 1 volt and a current density of 120 amperes per square meter of anode are employed in the electrolysis. In 1893, when I visited Lipine, the electrolytic plant there was producing from 500 to 600 kg. of refined zinc per day, the product assaying 99·93% Zn, 0·06% Pb and 0·01% Fe. The consumption of power was said to be approximately 30 h. p. per day.

Aside from the plant at Lipine, the only applications of the electrolytic refining of zinc that I am aware of have been for the treatment of the zinc crusts obtained in the desilverization of lead, which subject is of interest to the lead refiner rather than to the zinc smelter.

Recovery of Zinc from Waste Products.—The process of galvanizing iron and steel, in which a very large percentage of the production of spelter is consumed, results in a considerable quantity of waste products containing zinc, which are resold to the smelters for its recovery. These waste products are known as dross, sal-ammoniac skimmings and oxide skimmings (zinc ashes). The skimmings are removed from the surface of the galvanizing bath, the sal-ammoniac from one side and the oxide from the other. The dross is the ferruginous zinc, which collects as a pasty mass at the bottom of the pots, whence it is removed by means of a perforated ladle, at the end of each week, and slabbed in molds. If the spelter is high in lead a fourth by-product, viz., zinky lead, will accumulate in the bottom of the pot, but that is something which the galvanizer can guard against in his purchase of the metal.

In the sheet iron galvanizing practice in the vicinity of Pittsburg, Penn., the quantity of dross produced amounts to about 11·5%, on the average, of the spelter used; it assays 95 to 98% Zn. The sal-ammoniac skimmings amount to about 8% of the spelter used and carry 60% Zn. The oxide skimmings, or zinc ashes, are very small in amount; they assay 65 to 85% Zn. An interesting method for the treatment of such by-products, which was employed at a plant near Philadelphia, was recently described by Mr. J. Richards,[1] whose scheme included both the recovery of

[1] Journal of the Franklin Institute, CLI, 445 to 455; CLII, 59 to 63.

zinc from galvanized iron scrap, and the recovery of tin from tin plate scrap.

Tin and Galvanized Iron Scrap.—The method employed for the treatment of this material consisted in immersing the tin plate scrap in chlorhydric acid until the tin was dissolved, followed by an immersion in water, then in water with a little lime, then again in water, and finally for a moment in a solution of copper sulphate. The vats containing the various solutions were arranged in a semi-circle, commanded by a central crane; they were 6 ft. in diameter, 6 ft. in depth, and were sunk in the ground. The scrap was packed in a wooden cage, holding 200 lb., which was handled with the aid of the crane. The object of dipping the cleaned scrap in the solution of copper sulphate was to deposit upon it a thin coating of copper in order to protect the iron from rust, until after consolidation under a drop-press it could be reheated and forged into blooms. The tin was recovered from the chloride solution by dipping galvanized iron scrap in it, whereby metallic tin was precipitated by the zinc dissolved from the iron, zinc chloride being formed by the reaction. The precipitated tin was then collected and remelted, and the zinc chloride was sold for use as a disinfectant. About 600 lb. of tin were recovered from 10 tons of scrap. The cleansed iron sold for $10 per ton.

Waste Products from Galvanizing.—Sal-ammoniac skimmings, which consist chiefly of zinc oxide dissolved in ammonium chloride and double chloride of zinc and ammonium, were leached with hot water and steam, whereby the chlorides were dissolved. They could be recovered for further use as galvanizing flux by boiling down the solutions and crystallizing them out. The residuum from the lixiviation, consisting of zinc oxide, dirt and shots of zinc, was tumbled in a revolving barrel whereby the shots of metal were cleaned so that they could be remelted on the top of a pot of molten zinc. The zinc oxide which became crushed to a powder was marketable at $30 per ton with manufacturers of zinc paint. The average yield of such skimmings from galvanizing was 20% spelter, 30% ammonium and zinc chlorides, 35% zinc oxide and 15% iron scale and dirt.

Zinc Dross.—The zinc dross was treated by melting and introducing into the molten metal (below the surface thereof) a mixture of sulphur, potatoes and leather scrap. After about 15 minutes, action having been completed, a little aluminum was introduced in order to break up dissolved oxides. After permitting to stand for a short time, pure metal could be ladled off, most of the impurities of the dross remaining at the bottom of the pot. A sample of 40 carloads of spelter obtained in this way assayed 98·35% Zn, 1·5% Pb and 0·15% Fe. Spelter refined a second time by the same process

contained from 99 to 99·75% Zn, the only impurity being lead. A third refining failed to raise the percentage of zinc higher than 99·85. Scrap zinc could be treated in the same way. The aluminum was most successfully introduced in the form of an alloy containing 98% Zn and 2% Al.

Zinky Lead.—The zinky lead which collects at the bottom of galvanizing baths was treated by remelting in a wide pot at a temperature only just above the melting point of lead. An iron ring, 2 ft. in diameter and 6 in. in depth, was placed on the surface of the metal, and the zinc was skimmed off within the ring as fast as it rose to the top in a pasty condition. The lead still contained about 2% Zn after this treatment; a further elimination of the zinc would be of course a simple matter by the ordinary refining process, which is employed in connection with the desilverization of lead. The impure lead found a ready market, however, for the manufacture of weights, counterbalances and for use as ballast, etc.

XIV.

CADMIUM AND ITS RECOVERY.

PHYSICAL PROPERTIES.—Cadmium is a metal closely allied to zinc, with the ores of which it frequently occurs. It is tin-white in color, malleable and ductile, and brilliant in luster when fresh, but becomes dull on exposure to the air. Its malleability and ductility are such that it can be beaten out into thin foil and drawn into wire. When heated to about 80° C. it becomes very brittle and may be powdered easily, resembling zinc in this peculiar property. It breaks under a gradually increasing strain with the characteristic fibrous fracture of a soft, tough metal. It is harder than tin, but softer than zinc; when deposited electrolytically its hardness is such that a weight of 4·5 g. on a diamond point is required to produce a scratch.[1] Like tin, cadmium gives out a peculiar crackling sound when bent.

The specific gravity of cast cadmium is 8·604, which is increased to 8·694 by hammering. It crystallizes in octahedra, differing in this respect from zinc, which takes rhombohedral forms. Its melting point is 316° C. according to Wood, 320° C. according to Rudberg, and 355° C. according to Wagner. Its boiling point is 746·2° C. according to Becquerel and 860° C. according to Deville and Troost. Recent determinations by D. Berthelot[2] have shown that the true boiling point is between those figures, three experiments with an extreme range of eight degrees giving a mean value of 778° C. Berthelot's determinations were made at the same time and in the same manner as his determinations of the boiling point of zinc, and are without doubt more reliable than any of the previous determinations. In boiling cadmium, an orange-yellow or yellowish-brown vapor is given

[1] According to this scale of hardness, nickel electroplate is rated at 10; antimony electroplate at 9; palladium, deposited bright, at 8; chromium deposited on copper at 7; an alloy of 60·5% cadmium and 39·5% silver at 5; cadmium, deposited bright, at 4·5; and burnished silver at 4. These figures, which are reported by Sherard Cowper-Coles in the Electrical Review (London), XLIII, 587, Oct. 21, 1898, represent the number of grams weight on a diamond point required to produce a scratch.

[2] Comptes Rendus, CXXXI, vi, 380 to 382; Journ. Soc. Chem. Ind., Oct. 3, 1900.

off. This vapor decomposes water with the formation of cadmium oxide and the evolution of hydrogen. When cadmium is heated in the air to redness, it burns to a yellowish-brown oxide, CdO.

The specific heat of cadmium is 0·05669 (Regnault) or 0·0576 (Dulong and Petit). Its electrical conductivity is 22·10 or somewhat lower than that of zinc.[1] Its thermal conductivity does not appear to have been determined. Its atomic weight is 112·3; its atomic volume 12·9.

ALLOYS.—Cadmium alloys with many of the heavy metals and amalgamates with mercury. The alloys with gold, platinum and copper are brittle; those with lead and tin are malleable and ductile. With silver, cadmium forms an interesting series of alloys which promise to be of considerable use in the arts. The alloy of 70% Cd and 30% Ag has a remarkable structural formation, which causes it to split in a peculiar manner when it is rolled. A similar occurrence has been noted by other observers in the case of the alloy with 66⅔% Cd and 33⅓% Ag. On the other hand the alloy of 66⅔% Ag and 33⅓% Cd is said to be very tenacious. Cadmium also forms alloys with aluminum. Experiments with one of these for military purposes created a largely increased demand for cadmium in 1897, which led temporarily to an important rise in the value of the metal, the average for that year being $2·70 per kg. against about $1·50 ordinarily.

When alloyed with lead and bismuth cadmium has the peculiar effect of still further lowering the melting points below the mean of those of the constituents, which property makes it valuable for the preparation of fusible metals. Rose's metal, which is composed of 420 parts of bismuth, 207 parts of tin, and 236 parts of lead, corresponding to the formula Bi_2Sn_2Pb, melts at 93·75° C., but by the addition of 8 to 10% of cadmium its melting point is brought down to 75° C. An alloy, known as Lipowitz's metal, containing eight parts of lead, 15 parts of bismuth, four parts of tin, and three parts of cadmium becomes pasty at 55° C. and completely liquid at a little above 60°.[2] It is silver-white in color and 9·4 in specific gravity. The melting point of solder containing 37% Pb and 63% Sn is reduced to 136° C. by the addition of 8% Cd and to 132° C. by the addition of 25% Cd.[3]

Cadmium forms several amalgams with mercury. Those consisting of 50% Cd with 50% Hg and 33⅓% Cd with 66⅔% Hg are remarkable for their malleability and cohesive power. These amalgams were formerly used by dentists for stopping teeth, but are not so employed now. Another

[1] Prof. H. E. Armstrong, in Encyclopedia Britannica, 9th ed., V, 526, gives the electrical conductivity of cadmium as 23·72 at 0°C.

[2] Lipowitz, Dingler's Polytech. Journ., CLVIII, 376.

[3] Hauer, Dingler's Polytech. Journ., CLXXVII, 154.

amalgam composed of 78·25% Hg and 21·75% Cd, corresponding to the formula Hg_2Cd, is hard and brittle.[1]

CHEMISTRY.—The chemical behavior of cadmium is in most respects similar to that of zinc. It is always bivalent. It is soluble in sulphuric, chlorhydric, nitric, sulphurous and acetic acids. Dilute nitric acid is the best solvent. Zinc will precipitate cadmium from these acid solutions in a dendritic form, like the well known lead tree. Cadmium can also be deposited from its solutions by electrolysis. Cadmium decomposes water when its vapor is allowed to interact with steam at a red heat. When exposed to damp air cadmium becomes covered rapidly with a film of suboxide, but as in the case of zinc this oxidation is only superficial, the crust protecting the subjacent metal from further change. In oxygen, air, and chlorine cadmium burns in the same manner as zinc and magnesium do.

The most important chemical compounds of cadmium are the chloride, the sulphide, the sulphate, the carbonate, the hydrate and the cyanide.

Cadmium chloride, $CdCl_2$, is prepared by dissolving the metal, or its oxide, in chlorhydric acid. Like zinc chloride this is a volatile substance at comparatively low temperatures.

Cadmium sulphide, CdS, is one of the most characteristic compounds of the element. It is a beautiful yellow substance which is precipitated by hydrogen sulphide from a solution of a cadmium salt; the solution should be only moderately acid. The ability to precipitate the sulphide from acid solutions is another distinguishing characteristic of cadmium; zinc sulphide does not separate except from neutral or alkaline solutions. The shade of the precipitated sulphide of cadmium varies according to the conditions of precipitation. According to Niederländer[2] cadmium sulphide of a pale yellow color is produced from solutions of the chloride or sulphate when the current of hydrogen sulphide is interrupted after half of the metal has been precipitated, when the gas is allowed to come into contact as little as possible with the precipitate, and when the latter is washed with hot water. A dark yellow precipitate is produced by completing the reaction with constant stirring and continuous heating during the operation. A 10% solution is best suited to the production of both the pale and the dark yellow sulphide. Orange yellow sulphide is produced when hydrogen sulphide is passed into a nearly boiling solution containing 20% of cadmium chloride and 5% of free chlorhydric acid, the precipitate being stirred constantly throughout the process. Cadmium sulphide can be produced also by precipitation with an alkaline sulphide.

[1] Encyclopedia Britannica, 9th edition, IV, 628.
[2] Chem. Ztg., 1893, No. 82.

Cadmium sulphide is insoluble in dilute acids, but is soluble in concentrated chlorhydric acid even at ordinary temperatures. At white heat it melts, solidifying upon cooling in lemon-yellow scales of a micaceous structure. When the precipitated sulphide is heated in hydrogen it is decomposed, forming cadmium vapor and hydrogen sulphide, which if cooled together reunite in crystalline form. Cadmium sulphide is employed as a constituent of yellow pigments. These are quite permanent, unlike the lead and chromium yellows which are subject to more or less discoloration when exposed to atmosphere in which hydrogen sulphide is present.

Cadmium sulphate, $CdSO_4$, is obtained by crystallization in a manner similar to that employed for zinc sulphate, but it is not analogous to the latter in composition inasmuch as the crystallized sulphate of cadmium is represented by the formula $3CdSO_4+8H_2O$. The anhydrous sulphate, however, is simply $CdSO_4$, corresponding to $ZnSO_4$. Cadmium sulphate is used in medicine to a small extent; it has no industrial employment.

Cadmium Carbonate, $CdCO_3$.—According to Remsen, the neutral carbonate of cadmium, $CdCO_3$, is precipitated by soluble carbonates from acid solutions.

Cadmium oxide, CdO, is reduced to metal by carbon and carbon monoxide in precisely the same way as zinc; the temperature at which the reduction takes place is lower, however, than in the case of zinc. This difference enables a separation of the two metals to be made in a dry way and is the basis of the process employed for the commercial production of cadmium. The hydroxide of cadmium is soluble in ammonia, but is insoluble in an excess of the carbonates of potassium, sodium or ammonium. Cadmium hydroxide differs also from zinc hydroxide in not having acid properties, wherefore it does not dissolve in the caustic alkalies, forming compounds analogous to the zincates. Cadmium oxide is normally a yellowish brown substance; consequently the presence of cadmium is objectionable in ore or spelter that is to be used for the manufacture of zinc white.

Cadmium also forms a suboxide, Cd_2O, which is a green powder; on treatment with acids it is decomposed into metallic cadmium and CdO, the latter combining with the acid used.

Cadmium cyanide, $Cd(CN)_2$, is formed as a white precipitate when potassium cyanide is added to a fairly concentrated solution of a cadmium salt. It dissolves in an excess of potassium cyanide in consequence of the formation of the compound $K_2Cd(CN)_4$. This salt of cadmium is employed especially for the electrolytic deposition of the metal (q. v.).

USES.—Cadmium is used chiefly in its metallic form for the preparation of alloys. In the form of its sulphide it is employed as a pigment, which

is too costly, however, for general consumption and is used chiefly by artists. Cadmium is also coming into use to some extent as a covering for other metals, upon which it is deposited electrolytically. Its inertia to chemical action as compared with zinc and brass renders it suitable for coating the terminals and connections of primary and secondary batteries; also for coating small shot for sporting purposes and steel bullet jackets, in the one case to prevent leading of the barrels and in the other to prevent corrosion of the steel. Alloys of silver and cadmium deposited electrolytically are finding considerable employment as described in a subsequent section of this chapter.

Occurrence of Cadmium in Ores.—Cadmium occurs in nature in the form of its sulphide, CdS, as the mineral greenockite or cadmium-blende, an orange yellow mineral of 4·5 to 4·908 specific gravity, which crystallizes in the hexagonal system and contains 77·6% Cd. This mineral is of rather rare occurrence and is of no commercial importance, the cadmium of commerce being obtained entirely from cadmiferous blende and smithsonite, especially the Upper Silesian, which are noteworthy for their tenor in cadmium. The latter is not, however, by any means so high as the 2% to 5% as is stated in many metallurgical text-books, at least not in the ore at present mined in Upper Silesia.

The subject of the cadmium tenor of zinc ores has been studied by Edmund Jensch in a most painstaking manner, with the result that but few zinc ores were found to contain more than 0·5% of that metal.[1] In 10 years investigation he failed to find an Upper Silesian ore with more than 0·3% Cd. I have summarized the results of Herr Jensch's analyses in the following table:

Source of the ore	Class	No. of Analyses	Highest	Lowest	Average
Samuelsglückgrube, near Beuthen, O. S.	Blende	5	0·124%	0·040%	0·095%
Grosserschacht, near Birkenhain, O. S.	"	2	0·130	0·116	0·123
Neu-Viktoriagrube, near Karf, O. S.	"	8	0·110	0·044	0·068
" " " " " "	"	3	0·259	0·198	0·230
Neu-Helenegrube, near Scharley, O. S.	"	3	0·160	0·124	0·143
" " " " "	Calamine	1	.....	.....	0·196
Aufschlussgrube, near Beuthen, O. S.	Blende	2	0·166	0·140	0·153
Neuhofgrube	"	1	.....	.....	0·172
Mariagrube, near Miechowitz, O. S.	"	2	0·240	0·224	0·232
Kramersglückgrube, near Beuthen, O. S.	Calamine	1	.....	.....	0·306
Karl Gustavgrube, near Tarnowitz, O. S.	"	1	.....	.....	0·028
Medardusgrube, near Stollarsowitz, O. S.	"	1	.....	.....	0·090
Kleinigkeitgrube, near Radzionkau, O. S.	?	6	0·022	0·004	0·012
Samuelsglückgrube	Calamine	1	.....	.....	0·194
Karolinewunschgrube	"	1	.....	.....	0·009

[1] Das Cadmium, sein Vorkommen, seine Darstellung und Verwendung, in Sammlung chemischer und chemish-technischer Vorträge, III, vi, pp. 201 to 232 (1898).

The average cadmium tenor of all the Upper Silesian ore examined was only 0·102%.

A large number of analyses of blendes from many parts of Europe showed the presence of cadmium in most of them, but generally only in minute quantity. The richest was a sample of 208 tons of black blende loaded at Abo, Finland, in 1890, which contained 0·46% Cd, together with 34·43% Zn and 6·22% Pb. Large quantities of Swedish blende showed 0·2 to 0·4% Cd., while the blende from the Cantabrian coast (Santander) of Spain is known to contain 0·3 to 0·4% Cd. Apparently these Spanish ores and those from Sweden and Finland are the richest in cadmium of what is marketed in Europe at the present time.[1]

METALLURGY.—The methods of cadmium winning are simple, depending on the difference in temperature at which it and zinc are reduced and volatilized. In roasting blende containing cadmium sulphide the latter is oxidized together with the zinc. Either because cadmium oxide is volatile at the temperature which prevails in the roasting furnace or because of the reduction of the oxide by the carbon and unburned carbon monoxide in the gases of combustion, and the volatilization of the metal so reduced, there is a large loss of cadmium in the roasting furnace, as has been pointed out by Jensch,[2] who cites an instance wherein an ore assaying 0·110% Cd. before roasting (i.e., about the average of the Upper Silesian zinc ore) showed only 0·042% after roasting, a loss of 61·8%. The dust collected in the flues connected with the roasting furnaces always contains cadmium, though seldom more than 2%. A sample of such flue dust from the Hohenlohehütte assayed 1·64% Cd.; from the Godullahütte, 2·02%; from the Beuthenerhütte, 0·35%; and from the Kunigundehütte, 0·22%. About 40% of the cadmium in roasting furnace flue dust exists as sulphate soluble in water.

The cadmium oxide remaining in the roasted ore is reduced in the retorts of the distillation furnace by the carbon and carbon monoxide, according to the same reactions by which the reduction of the zinc is effected. The cadmium distils over before the zinc, however, and burns at the noses of the condensers with a characteristic brown flame and smoke before the bluish-green zinc flame appears; its presence in the ore is thus made known. Some of the cadmium vapor is collected in the condensers and prolongs,

[1] These analyses refer to the ores as marketed. If the cadmium in the Finnish ore quoted were contained entirely by the blende, the proportion in the picked mineral would have been nearly 1%. The high cadmium assays that have been reported are probably of pure minerals. Spanish calamine is said to have shown 2 to 5%; blende from Przibram 2 to 6%; blende from Eaton, N. H., 3·4%.

[2] Loc. cit.

going partly into the spelter and partly into the blue powder, from the latter of which it may be recovered. If cadmium is to be saved, the first deposit in the condensers and prolongs is collected apart and reserved for further treatment, usually that which is deposited during the first two hours of the distillation. This may contain 8% Cd, but the percentage is generally less than six. The zinc dust subsequently collected is unlikely to contain more than 1·2 to 2% Cd. In Upper Silesia where the Dagner and Kleeman condensers are now generally used, wherewith the escaping fume is conducted to a collecting chamber through one large pipe, the cadmium which escapes condensation is burned to oxide and is deposited in the part of the pipe nearest the furnaces, whence it is removed periodically or when the collecting chambers are cleaned out. This fume may contain as much as 5% Cd, but in general it is collected so as to assay from 3 to 4%.

Although the average percentage of cadmium in the Upper Silesian ore is very small, probably not more than 0·1%, the tonnage of ore smelted is so great that its total cadmium contents are large. The losses in the process of smelting are, however, enormous. It has already been mentioned that something like 40% is lost by volatilization during the roasting, when the ore treated is a sulphide, and probably only a small part of that loss is recoverable in the flue dust. In the subsequent distillation of the roasted ore further large losses are experienced through failure to condense the metal, which is more volatile and therefore more difficult to condense than zinc, through its going into the spelter and thereby becoming lost as cadmium, and through absorption of cadmium vapor by the clay of the retorts. In a previous chapter of this treatise it was pointed out that in Upper Silesia about 23,120 kg. of cadmium are probably lost annually in the last manner alone. The maximum production of cadmium in any one year was 15,527 kg. Assuming that the Silesian spelter contains an average of 0·025% Cd, the annual production carries about 25,000 kg. of cadmium. The cadmium recovered and sold as such, together with that which is marketed as spelter and that which is supposed to be absorbed by the retorts, account for about 65,000 kg. per annum, while if the ore smelted in Upper Silesia contained an average of only 0·05% Cd its total content would be upward of 250,000 kg. per annum.

Fractional Distillation.—The cadmium is recovered from cadmiferous blue powder by subjecting the latter to repeated distillations. The losses to be expected in those processes were determined by Edmund Jensch[1] in a series of experiments at the Kunigundehütte and reported in the following tables:

[1] Das Cadmium, sein Vorkommen, seine Darstellung und Verwendung.

Day	Kg. Blue Powder	Assaying, per cent.			Containing, grams		
		Zn	Pb	Cd	Zn	Pb	Cd
1.....	37·5	83·00	5·49	4·20	31,125	2,059	1,575
2.....	35·0	82·98	5·36	4·15	29,043	1,876	1,452
3.....	40·0	84·16	5·51	4·16	33,664	2,204	1,664
4.....	37·0	85·11	5·34	4·01	31,490	1,976	1,484
5.....	37·5	84·12	5·12	3·88	31,545	1,916	1,455
6.....	37·5	85·10	5·16	4·05	31,812	1,930	1,492
7.....	44·5	85·00	5·11	3·98	37,825	2,274	1,771
Total.....	269·0				223,504	14,235	10,893

The distillation of the above zinc dust yielded a cadmium-zinc dust of the following quantities and composition:

Day	Grams	Assaying, per cent.			Containing, grams			Percentage of Yield		
		Zn	Pb	Cd	Zn	Pb	Cd	Zn	Pb	Cd
1.....	1995	47·70	0·25	49·87	948·9	49·9	995·0	3·46	0·24	63·05
2.....	1110	43·88	.....	45·90	487·0	.....	509·5	1·76	.....	35·10
3.....	870	54·55	tr.	44·34	474·5	.....	385·8	1·41	.....	23·20
4.....	1013	41·28	tr.	57·26	418·1	.....	580·0	1·32	.....	39·10
5.....	2600	40·21	0·01	50·19	1045·5	0·26	1305·0	3·31	.....	89·70
6.....	2312	52·11	tr.	47·63	1205·8	.....	1101·7	3·44	.....	73·80
7.....	2155	55·66	tr.	37·65	1189·5	.....	811·4	3·14	.....	45·80
Total....	12055				5769·3		5688·4		0·034	52·20

The irregularity in the results was attributed to failure in maintaining uniformly the temperature of the furnace. The quantity of cinder (reduction material) used and residuum produced in these experiments was as follows:

Day	Weight of zinc dust Kg.	Cinder			Residuum			
		Kg.	$\%H_2O$	Kg. dry	Kg.	% Zn	% Pb	% Cd
1.....	37·5	85	19·5	68·42	100	30·21	1·17	.....
2.....	35·0	85	19·5	68·42	110	25·44	1·34	0·23
3.....	40·0	86	19·5	69·23	112	29·49	1·45	0·15
4.....	37·0	89	21·9	62·39	92	32·65	1·86	0·12
5.....	37·5	82	19·7	65·85	98	30·95	2·03	0·15
6.....	37·5	85	20·5	67·57	100	30·49	1·90	0·74
7.....	44·5	81	20·0	64·80	92	34·94	2·01	0·21

The disposition of the lead and zinc was accounted for therefore as follows:

Day	Zinc					Lead			
	Charged grams	Content of residuum grams	Content of dust grams	Total accounted for grams	Per cent. lost	Charged grams	Content of residuum grams	Total accounted for grams	Per cent. lost
1.....	31,125	30,210	949	31,159	*1·50	2,059	1,170	1,170	45·60
2.....	29,043	27,984	487	28,471	1·94	1,876	1,474	1,474	21·50
3.....	33,664	33,029	475	33,504	0·50	2,204	1,624	1,624	26·32
4.....	31,490	30,038	418	30,056	4·90	1,976	1,701	1,701	13·92
5.....	31,545	30,331	1,046	31,377	0·54	1,916	1,989	1,989	*3·70
6.....	31,812	30,490	1,206	31,696	0·37	1,930	1,900	1,900	1·56
7.....	37,825	32,135	1,189	33,324	11·99	2,274	1,849	1,849	18·69
Total.	226,504	214,217	5,770	219,587	7·38	14,235	11,707	11,707	17·76

* Apparent gain.

In the treatment of the whole quantity of 269 kg. of zinc dust there was consequently a loss by volatilization and absorption in the retorts of 7·38% of the zinc, 17·76% of the lead and 47·8% of the cadmium. The residuum was considered as a zinc ore and was subjected to redistillation in the ordinary manner. The cadmium-zinc dust was redistilled with the following result:

Day	Distillate II.			Yielding Distillate III.			
	Weighing grams	Containing		Weighing grams	Assaying		Containing
		Zinc grams	Cadmium grams		Zinc %	Cadmium %	Cadmium grams
1.....	1,995	918·9	995·0	1,027	5·67	94·2	967
2.....	1,110	487·0	509·5				
3.....	870	474·5	385·8	386			363
4.....	1,013	418·1	580·0	2,790	....	99·89	2,790
5.....	2,600	1,045·5	1,305·0				
6.....	2,312	1,205·8	1,101·7	1,303	1·2	98·65	1,285
7.....	2,155	1,189·5	811·4				
Total.	12,055	5,769·3	5,688·4	5,506			3,405

The cadmium obtained as the result of the third distillation was extremely soft and malleable. Out of the original 10,893 g. there was won 3405=31·26%.

Old Method.—The cadmiferous zinc dust used to be worked in Upper Silesia for cadmium by fractional distillation in precisely the same manner as indicated above. The blue powder was mixed with coke-breeze in the proportion of 0·66 cu. m. of the latter to 100 kg. of the powder and was distilled at red heat in an ordinary retort whereby there was obtained a

portion of the zinc condensed in liquid form and a blue powder enriched in cadmium. The distillation of the first blue powder required 12 hours and three charges were worked off before the residues were removed from the retort. The enriched dust obtained from the second distillation was mixed with charcoal and subjected to a third distillation at red heat in small cylindrical cast iron retorts, provided with sheet iron condensers and heated by the waste gases of the large distillation furnaces. Metallic cadmium in liquid form was obtained as a product of this distillation, the time required for which was 12 hours as in the case of the second distillation, and also as in the case of the latter the residues were removed only after every third charge. The crude cadmium thus obtained was remelted and cast in thin bars (Stängen).

At Engis, in Belgium, cadmium used to be produced in a similar manner, the blue powder averaging 1·5 to 1·6% Cd [1] being mixed with fine coal and distilled in furnaces of the ordinary Belgian type, but arranged with three rows of small cast iron retorts, fitted with cast iron condensers and sheet iron prolongs. This distillation yielded a blue powder containing 6% of cadmium. A furnace with 11 retorts would work off 100 kg. of the first powder in 12 hours and yield from 13 to 14·5 kg. of the enriched powder. The enriched powder was redistilled in the same furnace, 13 to 14·5 kg. being reduced in 12 hours in four retorts. The cadmium condensed as liquid metal in the second distillation was drawn off at intervals of one hour in order that it might be contaminated as little as possible by the iron of the condenser. The residue from the distillation assayed 0·3% Cd. Of the cadmium contents of the original blue powder, 30·12% was recovered in the form of metal,[2] 21·17% being left in the residues and 48·71% being volatilized. Three grades of cadmium were produced, the poorest amounting to approximately 50% of the total; this metal was sufficiently pure to be bent readily. The second grade of metal contained 75% Cd. and could be bent only with difficulty, but yet without breaking. The third grade of metal contained only 40% Cd and was very brittle. The impure metal could be refined by repeated fractional distillation.[3] Cadmium is no longer produced in Belgium.

Modern Method.—The cadmium produced in Upper Silesia at the present time is recovered almost entirely from the cadmiferous flue dust collected from the canals into which the fumes from the Dagner and Kleeman con-

[1] This was obtained from blende assaying 0·13 to 0·21% Cd, which after roasting contained only 0·02 to 0·03% Cd.

[2] This percentage of recovery is almost identical with that effected in Upper Silesia in the treatment of similar material. Refer also to Jensch's experiments, pp. 587, 588.

[3] Städler, Journ. f. Prakt. Chem., 1864, XCI, 359; Dingler's Polytech. Journ., CLXXIII, 286.

densers are discharged. Herr Otto Saiger, director of the works of Georg von Giesche's Erben, in Upper Silesia, which furnish about two thirds of the cadmium produced in that province, equivalent to saying two thirds of the world's production, has furnished the following notes as to the present practice there:[1]

"Cadmium being more volatile than zinc passes over first in the distillation of zinc ores which contain it, and escaping condensation, enters the large pipes leading away the fumes from the zinc furnaces employed in Upper Silesia, where it is deposited as oxide, this deposit being richer in the parts more remote from the furnaces. This condense ume, which at the Wilhelminehütte averages 3 to 4% Cd, and sometimes rises to 5%, having

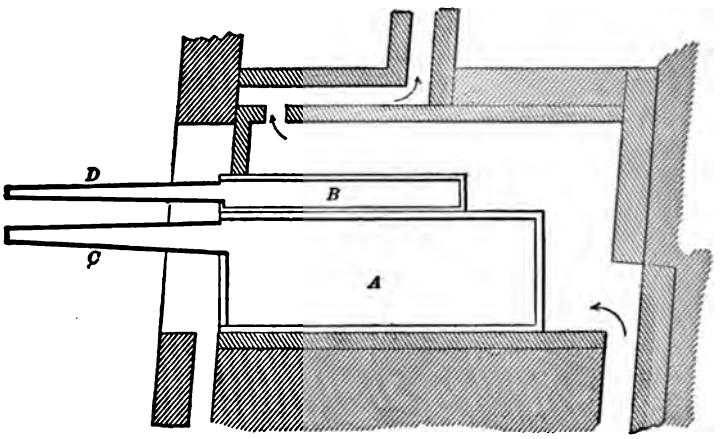


FIG. 393. RETORTS FOR CADMIUM DISTILLATION.

been collected by itself, is mixed with fine coke and charged into the retort *A*, shown in the accompanying engraving, of a special furnace. This retort, or muffle, which is 1·8 m. in length, takes about 115 kg. of the cadmium-bearing oxide, mixed with about 17 kg. of coke. It is charged about eight o'clock in the morning and the distillation is finished about six o'clock the next morning. The temperature in the muffle is maintained at a medium red glow, whereby only the cadmium oxide is reduced. The metal collects in the sheet iron condenser *C*, partly in the form of powder. That which is molten is so pure that it requires only a remelting before casting into the small sticks (Stängen) which are known in the trade. The powder is collected by itself, and every three days is reduced in the smaller retort *B*,

[1] The Mineral Industry, VII.

the temperature of which is maintained somewhat higher than that of *A*. The metal obtained from the condenser D is over 99·5% pure and is likewise cast into sticks. The residual zinc oxide from the retort *A* is returned to the zinc distillation furnaces."

The residue from the distillation in the retort *A* assays about 30·55% Zn and 1·36% Cd.[1] Being recharged with a fresh lot of zinc ore its cadmium content, or a part of it, is redeposited in the dust flues and therefore is not entirely lost.

COMPOSITION OF COMMERCIAL CADMIUM.—Metallic cadmium in order to be marketable must contain at least 99·5% Cd. The Silesian product generally assays between 99·50 and 99·75% Cd. A sample of metal from the Kunigundehütte assayed 99·8% Cd and 0·005% Fe. A sample from another Silesian works showed 99·65% Cd and 0·01% Fe. A sample of so-called "chemically pure" cadmium showed 0·022% Fe, 0·004% Cu and 0·011% Zn; another, 0·020% Fe, 0·006% Cu, and 0·017% Zn.[2]

STATISTICS OF CADMIUM PRODUCTION.—The production of cadmium in Upper Silesia since 1881 has been as follows:[3]

Year	Quantity, Kg.	Value, marks	Value, Per Kg.	Year	Quantity, Kg.	Value, marks	Value, Per Kg.
1882	3,521	.....	.....	1892	3,200	11,400	3·56
1883	2,419	.....	.....	1893	5,285	21,844	4·13
1884	2,775	.....	.....	1894	6,847	35,266	4·99
1885	3,276	.....	.....	1895	6,847	38,038	5·56
1886	4,964	36,599	7·47	1896	10,666	81,738	7·66
1887	7,321	48,497	6·62	1897	15,527	176,627	11·37
1888	4,796	22,855	4·22	1898	14,948	124,650	8·34
1889	5,138	19,503	3·79	1899	9,840	65,312	6·64
1890	4,158	14,610	3·50	1900	13,533	82,037	6·06
1891	2,849	10,182	3·52	1901	13,144	83,003	6·31

ASSAY.—Cadmium is readily detected by the reddish brown deposit which is formed on charcoal when the sample under examination is volatilized before the blow-pipe, whereas zinc under the same conditions gives a deposit which is bright yellow while hot, becoming white upon cooling. In quantitative analysis it is separated from zinc by precipitation from an acid solution by means of H_2S. The precipitated sulphide may be redissolved in chlorhydric acid and titrated with a standard solution of potassium ferrocyanide, or it may be precipitated as carbonate with Na_2CO_3, and after calcination weighed as oxide. Cadmium may also be determined electrolytically.

[1] Jensch, Des Cedmium, sein Vorkommen, seine Darstellung und Verwendung.
[2] G. Lunge, Chemische-technische Untersuchungmethoden, 1900.
[3] Statistik des Oberschlesischen Berg-und Hüttenwerke.

E. H. Miller and R. W. Page recently investigated several methods for the quantitative determination of cadmium.[1] They found the electrolytic method to be very accurate if a large excess of potassium cyanide and the presence of other salts be avoided. The carbonate method is the most troublesome and the least accurate. A very accurate method is the precipitation of cadmium from a neutral solution (cold) by a large excess of di-ammonium hydrogen phosphate and either weighing the precipitate as $CdNH_4PO_4,H_2O$ on a tared filter (after drying at 100 to 103° C.), or dissolving it in dilute nitric acid and igniting to pyrophosphate.

ELECTROLYTIC DEPOSITION OF CADMIUM.—The electro-deposition of cadmium is performed comparatively readily under the same conditions as are necessary for the electro-deposition of zinc. Borchers has found, however, that current densities of 60 to 100 ampères per square meter, which give an unsatisfactory deposit of zinc, are capable of yielding a good and useful deposit of cadmium. Alloys of silver and cadmium can also be deposited electrolytically and of late years have been employed to a considerable extent for various purposes. Smee was one of the first to deposit cadmium electrolytically. He obtained tough deposits by adding ammonia to a solution of cadmium sulphate and dissolving the precipitate in a very small excess of ammonia; he was unable to obtain good deposits from solutions of cadmium sulphate and cadmium chloride. In 1849 Messrs. Woolrich and Russell, of Birmingham, England, took out a patent for the deposition of cadmium. They prepared a solution of cadmium nitrate by dissolving the metal in nitric acid diluted with six times its bulk of water at a temperature of 80 to 100° F. Cadmium carbonate was precipitated from the solution by means of a solution of sodium carbonate. The precipitate was washed three or four times with warm water. It was then dissolved in a solution of cadmium cyanide of which 10% excess above the quantity required to dissolve the precipitate was employed. This solution was subjected to electrolysis, the best strength being found to be 6 oz. troy of the metal to the gallon. The temperature of the bath was maintained at about 80 to 100° F. Bertrand claimed to have obtained white adherent coatings from a solution of the bromide slightly acidulated with sulphuric acid. According to Sherard Cowper-Coles a strong solution of the double salt of cyanide of cadmium and potassium is one of the best, since the metal can be deposited rapidly from it and in a bright form, while anodes of cadmium dissolve very freely.

Uses of Silver-Cadmium Alloys.—Since 1895 alloys of cadmium with a small percentage of silver have been employed to a considerable extent in

[1] School of Mines Quarterly, 1901, XXII, iv, 301 to 308.

England for coating the steel parts of such machines as bicycles, etc., and a silver-cadmium alloy containing 7·5% of cadmium has been somewhat extensively used for plating domestic articles. Such alloys have been found to withstand the tarnishing influences of the atmosphere much better than pure silver or standard silver containing 7·5% of copper. The properties of these alloys and the manner of depositing them electrolytically have been described by Mr. Cowper-Coles, from whose paper these details have been taken.[1] A silver-cadmium alloy tested with a Thompson galvanometer was found to be electro-positive to nickel. Consequently if the alloy be used for coating steel and the underlying metal be at any time exposed by abrasion or otherwise, the galvanic action between the silver-cadmium coating and the steel is less intense than between the nickel and the steel. In order to determine whether silver-cadmium formed a more protective covering than nickel plate, samples of iron coated with cadmium-silver and nickel respectively were subjected to corroding influences. It was found that the nickel-plated iron in all cases rusted badly in a very short time, while the silver-cadmium coated iron was but little attacked. The difference in behavior was attributed to two causes: firstly, the nickel plate being porous permitted the air and moisture to act on the iron, but that did not happen with the silver-cadmium coating, which appeared to be almost impervious; secondly, the silver-cadmium coating was found to be electro-positive as compared with nickel, to the extent of a difference of more than 0·25 volt.

Silver-Cadmium Plating.—The process which has been used chiefly for depositing cadmium-silver alloys consists in preparing the electrolyte by dissolving cadmium and silver cyanides in potassium cyanide the proportions of the metals being varied according to the nature of the deposit that is sought. In order to obtain deposits of 10 to 80% of cadmium it is found necessary to have the ratio of the silver and cadmium in solution from 1:4 to 1:7, the best results being obtained when the quantity of cadmium in solution is from 3 to 4 oz. per gallon,[2] and the quantity of silver not less than 8 dwts., or more than 25 dwts. The weaker the solution the smaller must be the current density employed, and in order to keep the bath from becoming exhausted the anode surface should be greater than the cathode surface and sufficient free cyanide always present in the bath to dissolve the cadmium and silver cyanide formed on the anodes. The addition of carbonates of the alkali metals is found to reduce local action, due probably to the nascent liberated metal. Observation of the color and

[1] Electrical Review (London), XLIII, 587, Oct. 21, 1898.

[2] This refers undoubtedly to the British imperial gallon, which is equivalent to 4·54 liters and 10 lb. of water. The troy ounce is equivalent to 31·1 g. and 1·097 ounces avoirdupois.

general appearance of the deposit on a test plate is found to give full control over the composition of the alloy deposited, an experienced plater being able to estimate the composition within 1%. As the nature of the deposit varies with the current density it can be regulated by altering the latter. The differences between the two metals play an important part. In the first place there is an electromotive force between the two metals independent of the back electromotive force of the whole electrolyte; furthermore a current that will deposit 402·418 kg. of silver will deposit only 2·079 kg. of cadmium; while there may be a considerable difference between the resistances of the respective salts in solution. If a bath be made up containing a certain number of ounces of metal per gallon, and deposits are obtained from different current densities, the percentage of silver obtained in the deposit will be different in nearly every case. Consequently careful experiment is necessary in order to determine the proper current density and other conditions necessary to produce a deposit of the desired character.

Solutions containing 2 or 3 oz. of metal per gallon and a large percentage of cadmium give with the small current densities which alone are permissible deposits varying from 40 to 100% of silver, whereas solutions containing 8 to 9 oz. of metal per gallon give 80 to 90% silver with a very wide range of current density. For any given solution the percentage of the electro-positive metal increases with the current density, but not in simple proportion. If the solution be a weak one, patchy deposits are the result of too high current densities, the metals appearing to separate out, although the deposit may be smooth. The proportion between the more electro-negative metal and the more electro-positive metal deposited at a given current density can be varied by stirring the solution. Movement of the cathode in the bath will temporarily alter the deposit, and if the bath is weak and the anodes are not dissolving properly it will greatly alter the resistance and consequently the current density.

Cowper-Coles summarizes briefly the conditions in saying that for every composition of the solution there is a definite current density to be used to produce a certain deposit, and if the bath is to be maintained in working condition for any length of time, the anode must be of the same composition as the deposit which is to be produced. The ratio of cadmium and silver in solution is dependent on the density of the solution, different proportions of the two metals being necessary if the solution is very dense. It is desirable, however, to use a strong solution, because a greater current density can be employed and consequently the alloy can be deposited more quickly. Stirring the solution permits the use of still higher current densities, but the proportions of the metals in the bath must be altered accordingly.

XV.

COST OF PRODUCING ZINC.

The principal items of expense in smelting zinc ore are labor, coal and clay. The cost of smelting will therefore depend chiefly upon the consumption of those things and their market value, but it will depend also in no slight degree upon the character of the ore which is to be smelted; or rather it should be said that given the same plant the consumption of labor, coal, clay, etc., will vary with the character of the ore. To what extent such variation will be experienced it is difficult to predicate in general terms, because there are so many confusing and conflicting factors. The result is that the cost of smelting a specified zinc ore cannot be estimated in advance with the same precision as in the case of a lead or copper ore and we are obliged to fall back upon the results of practical experience.

Relation Between Grade of Ore and Cost of Smelting.—It is well known that the cost of smelting a ton of zinc ore increases as the grade of the ore decreases. There are three reasons why that should be so: (1) The lower the percentage of zinc the higher is the percentage of gangue, and as a general thing the higher the percentage of objectionable impurities in the gangue, which *per se* enhance the cost of smelting; (2) the lower the percentage of zinc and the higher the percentage of gangue, the lower will be in many cases the average specific gravity of the ore, the greater its bulk for a given weight and the less the weight that can be smelted with a given plant; (3) the lower the percentage of zinc the higher will be the loss of metal in smelting. It will simplify the consideration of this subject if the effect of corrosive and contaminating impurities such as iron and lead be ignored for the present, and the case of ores consisting only of zinc oxide and silica, of which the latter may be regarded as inert, be discussed alone.

The specific gravity of zinc blende may be taken as four, making the weight of a cubic foot of the solid mineral 250 lb. The weight of a cubic foot of finely crushed ore would be from 137·5 lb. to 150 lb. according to the fineness of the crushing and conditions of the measurement. A cubic foot of Joplin concentrates, dry and assaying 58% Zn, actually weighed 130 lb. A cubic foot of sand or crushed quartz (sp. gr. 2·65) will

weigh from 80 to 100 lb. It is obvious that with different proportions of the two minerals the weight of a cubic foot of the mixture will be the less the higher is the percentage of the lighter mineral; in other words the lower is the percentage of the zinc. Moreover, the less the percentage of zinc in the case of a sulphide ore the less will be the loss of weight in roasting and the more the weight to be distilled. The capacity of a zinc smelting furnace is governed by the volume of its retorts. Consequently the more voluminous is the ore the less of it can be put into a given number of retorts and the less the capacity of a given furnace. The number of men required to manage the furnace remains the same, however, as does also the quantity of coal required to heat it.

Practically the zinc smelter is seldom confronted with the question of smelting ores of so simple a character. As a general thing the medium and low grades of ore contain lead and iron, instead of a quartzose gangue alone, which tend to increase their weight per cubic foot, or at least make it equal to that of an ore of high grade in zinc; or else the medium and low grades of ore may be calamine, which does not have to be crushed so finely as blende, and consequently is relatively less bulky. For example, roasted Missouri blende assaying 65% Zn crushed to 2 mm. size weighs approximately the same number of pounds per cubic foot as the raw calamine (hemimorphite) from the same district assaying only 42% Zn crushed to 4 mm. size, both of these ores being practically free from iron, lead and heavy gangue minerals. Although the two ores differ by 23% in their zinc tenor, it is possible to charge as much of one in a furnace of the same retort volume as of the other, and that is, or has been, done practically. In the case of a sulphide ore, moreover, if it be ferruginous the weight per cubic foot after roasting is not only likely to be higher than that of a non-ferruginous ore of a higher tenor in zinc, but also the loss of weight in roasting is likely to be greater, wherefore the cost of smelting if reduced to terms of the raw ore will be affected. Again there are blendes containing a gangue of calcite or dolomite, which in roasting lose more weight by the expulsion of carbonic anhydride than is replaced by absorption of sulphuric anhydride, and consequently may show a greater shrinkage than the highest grades of blende with a silicious gangue. The quantity of reduction material required by various ores, its character, and the percentage of between-products that will be made and have to be resmelted are also important considerations.

It is apparent therefore that although the grade of the ore has a bearing upon the cost of smelting, there are practically so many conflicting factors which also enter into the question that it is impossible to present any

empirical rules that will cover it. It is most convenient to refer the cost of distillation and the accessory processes to terms of the calcined or roasted ore, converting that figure into terms of the raw ore, if desired, by multiplying by the decimal representing the percentage of weight remaining after roasting or calcination. For example, if the cost of distillation per ton of roasted ore were \$7 and the roasted ore amounted to 85% of the weight of the raw ore, the cost per ton of the latter would be 0·85×\$7 =\$5·95.

It should be observed that in the discussion of this subject reference has been made solely to the relative cost of smelting different grades and kinds of ore by the same process, i.e., the Belgian. Generally speaking, that process ceases to be profitable when the ore does not contain at least 40% Zn, or about that figure. The Silesian process on the other hand permits the economical treatment, in the district where it is chiefly practiced, of ore assaying only 20 to 25% Zn, and even less, the cost per ton of ore being lower than for the treatment of 45 to 50% ore in Belgium. This difference appears to be due chiefly to the less cost per ton of ore for labor, clay and repairs and renewals, the consumption of coal and reduction material per ton of ore being about the same in each case.

Relation Between Grade of Ore and Recovery of Metal.—The difference in the percentage of zinc lost in smelting due to variations in the grade of ore can be calculated theoretically by making certain assumptions, which are based on the results of practice. In the present case it will be assumed that each unit of sulphur in the roasted ore will hold back one unit of zinc as sulphide, and that the quantity of zinc remaining in the residue as undistilled oxide will be equal to that retained as sulphide; also that the percentage of zinc condensed will be in all cases 90% of that which is distilled. In order to simplify the calculation it is assumed that the reduction material will be coke, free from volatile matter but containing 10% ash, that the quantity used will be 50% of the weight of the roasted ore and that one half of it will remain unburned in the residuum drawn from the retorts. For convenience the following figures are based on charges of 10,000 lb.:

Weight of charge	Assay of charge		Weight of coal charged	Weight of residue	Tenor of residue in zinc		Weight of zinc in roasted ore	Weight of zinc distilled	Percentage of zinc distilled	Weight of zinc condensed	Percentage of zinc in roasted ore condensed
lbs.	Zn. %	S %	lbs.	lbs.	lbs.	%	lbs.	lbs.		lbs.	
10,000	70·46	0·88	5,000	4,255	352	8·27	7,046	6,693·7	95·0	6,024·33	85·5
10,000	63·66	0·87	5,000	5,256	348	6·62	6,366	6,018·0	94·5	5,416·2	85·0
10,000	57·01	0·86	5,000	6,073	344	5·66	5,700	5,356·0	94·0	4,820·4	84·5
10,000	50·59	0·84	5,000	6,865	336	4·90	5,059	4,723·0	93.3	4,250·7	84·0

The above computations are based on the theory that an ore cannot be desulphurized below a certain point, regardless of its original tenor in sulphur, because of the cessation of chemical action (in this case oxidation) beyond a certain limit. For a similar reason the retort residues may be expected always to contain about the same quantity of undistilled zinc oxide. The result of this is that in the distillation of 1000 lb. of ore there will remain in the residuum approximately the same number of pounds of zinc, irrespectively of the quantity of the latter metal originally present in the ore distilled. It is obvious therefore that the loss of metal will increase in the same proportion as the grade of the ore diminishes.

The computations presented above show that in the distillation of ores of precisely the same character, but of different tenors in zinc, the percentage of zinc volatilized, condensed and recovered ought theoretically to decrease about 0·1 for each unit of decrease in the grade of the original raw ore. Actually the decrease will be greater than 0·1% per unit, for the reason that the percentage of volatilized zinc that will be condensed, which in the above computations has been assumed uniformly as 0·90 will also decrease with the grade of the ore. The latter phenomenon is attributable to the same theory which obtains with respect to the loss of zinc in the residuum, namely, that losses are approximately constant, irrespectively of the grade of the ore. Of the zinc which is volatilized losses are suffered through failure to condense because of zinc vapor which filters through the walls of the retorts, is absorbed in the latter, remains in the retort at the end of the distillation and burns at the noses of the condensers. Now it is known that the quantity of zinc absorbed by the retorts is about the same whether the ore be of high grade or low grade; also the zinc vapor remaining in the retorts at the end of the distillation, when the gas has insufficient tension to expel it, must necessarily be the same. Those losses will naturally increase in proportion as the grade of the ore decreases, and probably a similar result is experienced in the case of the losses by filtration through the retorts and burning at the ends of the condensers.

The table also illustrates the fact that, because of the less weight of the residues from a high grade ore, a high assay in zinc does not necessarily imply a greater actual loss than is experienced in a lower grade of ore, of which the residues carry a smaller percentage of zinc. Thus, the number of pounds of zinc remaining in the residues is about the same in the case of an ore assaying 70·46% Zn, yielding 0·43 of its weight as residue assaying 8·27% Zn; and an ore assaying 50·59% Zn, yielding 0·69 of its weight as a residue assaying 4·90% Zn. With respect to grade of ore, proportion of residue and zinc tenor of the latter, these figures represent approximately

the results of the practice in Kansas and Belgium respectively; the average grade of the ore smelted is in each case somewhat lower than 70·46 and 50·59% Zn respectively, and the average zinc tenor of the residua is also a little lower—certainly it is so in Belgium.

COST OF SMELTING REDUCED TO TERMS OF THE PRODUCT.—Although the cost of smelting per ton of ore varies with the grade of the latter, the difference is by no means so great as it is in the cost per pound of spelter produced, which, of course, increases enormously as the grade of the ore goes down. Thus if we had an ore assaying 60% Zn, or 1200 lb. per ton, which it cost $10 per ton to smelt, and another assaying 45% Zn, or 900 lb. per ton, which it cost $12 per ton to smelt, and if the recovery of metal from the high grade ore were 85·5% and from the low grade ore 84% the cost per pound of metal recovered would be as follows: $10÷1026=0·975c. per lb.; and $12÷756=1·587c. per lb. This difference must necessarily be offset by a correspondingly lower price paid for the zinc in the low grade ore.

Commercially the zinc smelter usually reduces his cost of smelting to terms of the product which he sells, namely, spelter; i.e., he reckons, for example, that the metal which he sells for 4c. per lb. costs him 3·6c., of which 2·6c. is for ore and 1c. for smelting. This is the custom in the United States at least. Looking at the matter from the business standpoint it is the simpler method. In a technical consideration of the cost of smelting with reference to metallurgical practice, on the other hand, it is simpler to consider it as coming to so much per ton of ore. The yield of metal from a ton of ore being stated, it is, of course, easy to convert one method of expression into the other. In smelting the average Joplin ore about one ton of metal is got from two tons of ore.

COST OF SMELTING.—The various items which enter into the cost of zinc smelting are conveniently summarized under the captions of (1) labor, (2) fuel, (3) reduction material, (4) clay for retorts and condensers, (5) repairs and renewals and sundry supplies, and (6) administration and general expense. These items naturally vary greatly at different places according to the character of the ore treated, the practice in smelting, the arrangement of the works and the cost of labor, fuel and other material. Of equal importance with the cost of smelting in determining the efficiency of a given plant, and the profit to be expected from it, is the percentage of metal recovered, which is also dependent upon the character of the ore treated, the practice in smelting and the arrangement of the works. The cost of smelting and the percentage of metal recovered are indeed closely related, inasmuch as in the treatment of the same ore in the same plant

the cost of smelting may be reduced at the expense of an increased loss of metal in the ore, and vice-versa.

Labor.—The largest single item of labor expense is always that which is involved in the operation of the distillation furnaces. Next in importance is that which is required for roasting the ore, if the latter be blende and if the roasting be done by hand, but the use of mechanical furnaces, especially in connection with gaseous fuel, reduces the cost of labor in roasting to a very low figure. The labor for making retorts and condensers constitutes a comparatively small item in the total, but the various other items, such as the handling and transportation of material about the works and the miscellaneous, general labor, amount in the aggregate to a rather large percentage of the total, varying greatly, however, according to the arrangement and size of the works.

In the smelting of 2000 lb. of raw blende in Kansas, including the roasting as well as the distillation and all handling of material, it may be considered roughly that there is required the labor of one man for 3·5 days if the fuel employed be coal and 2·5 days if it be natural gas, at an average wage of $1·80 per day. In Belgium and Germany the average wage appears to be about 70@80c. per day, but the efficiency of the labor is inferior to the American, although by no means so much as to offset the advantage which European smelters have with respect to the cost of labor.

Fuel.—By far the largest part of the consumption of fuel is due to the requirements of the distillation furnaces, the aggregate of all other items such as the generation of steam and power, the burning of retorts and condensers, etc., being comparatively insignificant. If the fuel employed be coal, the consumption varies within wide limits according to the calorific power of the coal, its character and the method of burning it. In the best practice in Euroope, the distillation of a ton of calcined or roasted ore is effected with as little as one ton of coal, and even less at certain works in Upper Silesia; the attainment of such a result is not to be expected, however, except with a fairly good grade of coal and the use of gas-fired, heat-recuperative furnaces. On the other hand, the heating of the retorts in direct-fired Belgian furnaces with an inferior coal, such as that of Southern Illinois, may require as much as four tons per ton of roasted ore.

Reduction Material.—The consumption of reduction material, i.e., the lean coal, cinder or coke which is mixed with the ore to be distilled, varies generally between 40 and 60% of the weight of the calcined or roasted ore.

Clay for Retorts and Condensers.—The consumption of clay for the manufacture of retorts and condensers amounts to about 0·10 ton per ton of roasted ore when the distillation is done in Belgian retorts. In Upper

Silesia the consumption appears to range between 0·07 and 0·13 ton per ton of calcined or roasted ore.

Repairs and Renewals.—This is always a large item in the cost of zinc smelting, chiefly because of the frequency with which the distillation furnaces must be relined and the comparatively high cost of effecting such a renewal. This will naturally vary a good deal according to the excellence of the original construction of the furnaces, the manner in which they are operated and the attention which is given them during the campaign. Repairs on other parts of the works, including the roasting and miscellaneous furnaces, steam and power plant, machinery, buildings, tramways, etc., will of course vary rather widely according to their character, first cost and the manner in which maintained. It is always difficult to make an accurate separation between the sundry supplies which are required for regular consumption, such as oil, waste, etc., for the engines and machinery and those which are required irregularly for repairs and renewals, wherefore it is better to lump all those items together unless a minute and elaborate system of bookkeeping be in effect.

Administration and General Expense.—Under this caption are included the salaries of executive officers, office expenses, postage, stationery, telegraph, telephone and traveling expenses, taxes, insurance, interest and discounts, legal expenses, etc. The aggregate amount will naturally vary a good deal according to the magnitude of the business and the character of the management. The cost of purchasing the ore supply and selling the product of spelter properly falls under this caption.

Recovery of Metal.—It has previously been pointed out how the percentage of metal recovered varies with the character of the ore smelted and its tenor in zinc. Some specific figures as to this may be found in Chapter XVII. Speaking generally it may be said that in the distillation of ore assaying about 25% Zn in Upper Silesia there is recovered as spelter from 70 to 75% of the zinc contents of the ore. Belgian and Rhenish smelters, distilling ore which assays 45 to 50% Zn, win from 85 to 90% of the metal. The Western smelters of the United States, distilling a higher grade and more docile ore, recover about the same percentage as the Belgians. In both cases the average results in the long run are probably nearer the lower figure than the higher.

The percentage of metal recovered is commonly referred to the ore distilled, i.e., to the ore after it has been roasted or calcined. The crude ore is subject to a further loss in roasting or calcining, which may amount to 2 to 3% of its zinc contents, but with proper plant and attention need not be anywhere near so great.

Inasmuch as in the treatment of the high grade ores which are smelted in the United States a difference of 1% in the proportion of metal recovered corresponds approximately to 50c. per ton of raw blende, at the ordinary price of spelter, the great importance of considering the matter of losses in smelting in connection with the cost of smelting is quite obvious.

Conditions in Kansas.—Zinc smelting in Kansas was until recently carried out contemporaneously under greatly different conditions, not however relating to the character of the ore which is rather uniform. One group of smelters employed coal as fuel; another had acquired the advantage of a natural gas supply. In each group some smelters roasted by hand and others by mechanical furnaces. At the present time the coal smelteries are mostly idle, having been unable to withstand the competition of the gas smelteries, but although the latter now establish the market conditions they continue to be of various degrees of efficiency. The difference in the cost of smelting in Kansas under various conditions is illustrated in the following table:

COST OF SMELTING CONCENTRATED BLENDE.

Item	Cost per ton of roasted ore *a*				Cost per ton of raw ore			
	Coal		Natural gas		Coal		Natural gas	
	Hand roasting	Mech. roasting	Hand roasting	Mech. roasting	Hand roasting	Mech. roasting	Hand roasting	Mech. roasting
Labor *c*	$7·70	$6·78	$5·47	$4·94	$6·62	$5·82	$4·70	$4·25
Fuel	*b* 2·63	*b* 2·63	.....	.....	2·26	2·26	.....	.....
Reduction Material *d*	·55	·55	·70	·70	·47	·47	·60	·60
Clay, for retorts, etc. *e*	·30	·30	·30	·30	·26	·26	·26	·26
Repairs, renewals and sundry supplies *f*	·87	·87	·87	·87	·75	·75	·75	·75
Total	12·05	11·13	7·34	6·81	10·36	9·56	6·31	5·86

a The roasted ore is assumed to be 86% of the weight of the raw ore (dry). *b* Reckoning a consumption of 3·5 tons of coal @ 75c. per ton, equivalent to a consumption of 3 tons per ton of raw ore. *c* Except labor charged to repairs and renewals. *d* Reckoning 0·5 ton per ton of roasted ore. *e* Reckoning 0·1 ton per ton of roasted ore. *f* Including all labor properly chargeable to account of repairs and renewals, besides expense of putting repaired furnaces in operation, etc.

According to the above estimates there is an apparent advantage of about $4 per ton of raw ore, equivalent to about 0·4c. per lb. of spelter produced, in favor of smelting with natural gas. It is to be observed, however, that the natural gas is assumed as costing nothing, which is an erroneous assumption, inasmuch as the expense of acquiring the land whence it is obtained, drilling for it, piping to the works and maintaining the supply must be charged to the cost of smelting; or redeemed out of the profits,

which is the same thing. In view of the uncertain character of a natural gas supply it is difficult to determine this factor, which is largely a matter of bookkeeping, but prudence decrees the most conservative consideration of it. There are certain smelters in the Kansas gas fields whose gas supply costs very little because of the small original outlay to secure it; there

STATISTICS OF THE AMERICAN ZINC SMELTING INDUSTRY.

	United States	Illinois	Indiana	Kansas	Missouri	Pennsylvania	Other States *a*
Number of Works	31	5	3	11	5	3	4
Capital invested: *b*							
Land	$2,243,876	$138,100	$7,250	$1,684,026	$25,500	$59,000	$330,000
Buildings	5,470,590	637,434	108,454	2,206,702	288,000	1,150,000	1,080,000
Machinery and tools	1,935,754	1,095,041	21,690	405,464	77,300	190,000	146,259
Cash and sundries	4,491,590	1,315,744	7,441	922,337	413,229	472,626	1,360,213
Total	14,141,810	3,186,319	144,835	5,218,529	804,029	1,871,626	2,916,472
Ore smelted	463,609	92,400	10,359	131,407	42,584	45,627	141,232
Dross smelted	12,578	2,427	.....	.....	.....	9,569	582
Spelter produced	131,546	46,759	4,359	54,516	18,107	4,781	4,024
Sheet zinc produced	17,723	17,723	.....	.....	.....	.....	.....
Sulph. acid produced	58,829	58,828	.....	.....	.....	.....	.....
Acid phos. produced	7,512	7,512	.....	.....	.....	.....	.....
Zinc oxide produced	37,557	.....	.....	.....	.....	9,644	27,913
Av. val. of spelter *c*	$108·49	$103·59	$102·24	$106·21	$111·10	$138·17	$133·09
Av. val. of sheet zinc *c*	140·80	140·80	.....	.....	.....	.....	.....
Av. val. sulphuric acid *c*	7·22	7·22	.....	.....	.....	.....	.....
Av. val. of zinc oxide *c*	72·39	.....	.....	.....	.....	80·84	69·46
Number of men: *d*							
Maximum	5,944	1,641	190	1,958	537	751	867
Minimum	4,473	1,476	140	1,186	433	593	645
Average	4,869	1,551	119	1,487	500	448	764
Total h. p. of engines	12,546	3,294	230	2,330	675	3,921	2,096
Cost of Production:							
Ore *e*	$10,995,846	$3,669,953	$326,191	$4,317,628	$1,503,670	$262,792	$915,112
Dross *e*	644,645	124,340	.....	.....	.....	506,242	14,063
Fuel	751,839	261,661	8,058	129,173	93,534	66,591	192,822
Rent of power	75,000	.....	.....	75,000	.....	.....	.....
Mill supplies	83,845	24,977	1,145	10,452	2,412	14,962	29,897
Other material	436,378	205,538	6,836	41,103	33,871	45,605	103,425
Freight	298,505	130,346	44,669	105,590	17,900	.....	.....
Labor	2,355,921	758,912	58,138	705 803	268,196	174,510	390,362
Salaries	440,200	222,022	7,650	60,800	36,880	13,669	99,179
Other gen. expense *f*	399,472	109,407	5,588	81,169	36,903	4,869	161,536

a Includes two works in New Jersey and one each in Virginia and Wisconsin. *b* Includes the value of land, buildings, machinery, tools and implements, and the live capital utilized, but does not include the capital stock of any corporations reporting. *c* Values at the works where produced. *d* Includes women and boys, of whom the number is insignificant; does not include proprietors, managers, superintendents, clerks and salesmen. *e* Cost delivered at the works. *f* Does not include selling expense, interest on capital or amortization of plant.

are others, owning the gas rights to a large area of land, to whom the gas they obtain from it must cost a good deal. Finally, in considering the relative economy of smelting with coal and with natural gas it must be borne in mind that the comparison is made with an antiquated and uneconomical practice of smelting with coal and not with a modern one. It is obvious that as between a natural gas smeltery and a coal smeltery equipped with gas-fired, heat-recuperative furnaces and a complete me-

chanical system of handling the coal, there would be much less in favor of natural gas smelting in the United States, where coal is obtainable so cheaply in many districts.

The statistics of the zinc smelting industry collected for the Twelfth Census of the United States are of considerable interest in this connection, although inasmuch as they are summarized by States, instead of by districts, and include the returns of the smelters who produce zinc white, sheet zinc, sulphuric acid and acid phosphate of lime besides spelter, it is difficult to make purely technical deductions from the data. The statistics collected by the Census cover the calendar year 1899; unfortunately the conditions of that year, in so far as the zinc smelting industry was concerned, were peculiar and abnormal, which fact must be borne in mind in considering its statistics. The latter, as reported by the Census, with the interpolation of some averages, are summarized in the accompanying table, in which the quantities of ore and dross smelted, and spelter, sheet zinc, oxide, sulphuric acid and acid phosphate of lime produced are reported in tons of 2000 lb., the average values being given per ton.[1]

Referring to the accompanying table, it is to be remarked that the great difference in the values of the spelter produced in the East and West, i.e., 6·65@6·90c. vs. 5·10@5·55c. per lb., is due partly to the superior quality of the former metal. Also it is to be borne in mind that there were wide fluctuations in price in 1899 and the period of the year at which the product was sold would make a great difference. In the latter half of the year most of the Western smelters were operating at a greatly reduced rate, and some closed their works entirely.

The production of the works of Indiana, Kansas and Missouri and the labor and fuel cost per ton of spelter, segregated according to the kind of fuel used, was as follows:

COAL.

	Spelter short tons	Wages per ton	Fuel per ton
Kansas and Southwestern Missouri	24,219	$15·53	$4·19
St. Louis District	14,151	14·06	5·68

NATURAL GAS.

	Spelter short tons	Wages per ton	Fuel per ton
Kansas	40,125	$12·03	$1·38
Indiana	4,359	13·34	2·99

[1] Compiled from Bulletin No. 124, entitled "Lead, Copper and Zinc Smelting and Refining."

The above works produced only spelter, no by-products being recovered. Under the caption of fuel, the coal and coke used for reduction material was doubtless included, although it was not so stated in the report. It was not stated what value, if any, was put on the natural gas consumed.

Conditions in Europe.—It is even more difficult to generalize with respect to the cost of zinc smelting in Europe than it is in Kansas, because the conditions differ so widely. In good practice in Belgium the distillation of 1000 kg. of roasted blende or calcined calamine requires about 1250 to 1500 kg. of heating coal, 400 kg. of reduction coal and 100 kg. of fire clay. In Rhenish Prussia and Westphalia the consumption of heating coal for the same grade of ore is probably not more than 1250 kg. on the average, being in some instances as low as 1000 kg. Assuming that the roasted blende were derived from raw ore assaying about 45% Zn and that the latter would lose 12·5% of its weight in roasting, 1·14 tons of blende would be required to produce one ton after roasting. A coal consumption of 1·9 tons per ton of roasted ore would be equivalent therefore to only 1·66⅔ per ton of raw ore; similarly the consumption of clay would be reduced to $100 \div 1{\cdot}14 = 87{\cdot}7$ kg. The consumption of 0·25 ton of coal in roasting a ton of ore would make the total $1{\cdot}667 + 0{\cdot}25 = 1{\cdot}917$ per ton of raw ore. The yield of spelter in the best practice might be as high as 90%, or $450 \times 0{\cdot}09 = 405$ kg. per metric ton of ore.

Expressed generally the European smelters have the advantage over the American of a rather better practice in distillation and a better grade of coal, leading to a higher percentage of metal extraction and lower consumption of coal, which is partly offset by the higher cost of the latter. They have also the advantage of much cheaper labor, which in its most important branch, i.e., the management of the distillation furnaces, is quite as efficient as the American. On the other hand, in those branches where the use of mechanical methods is possible, especially in roasting and the handling of material, the Americans have offset the higher cost of their labor by increasing its efficiency.

According to Ad. Firket[1] there were smelted in 1898 at the twelve works of Belgium 291,977 metric tons of ore and 11,770 tons of between-products, a total of 303,747 tons, which yielded 119,671 tons of spelter and required the consumption of 624,511 tons of coal. These figures refer chiefly to the distillation process, because in Belgium the roasting of blende and the calcination of calamine is done generally at different plants from the smelteries proper. Ignoring the consumption of coal in the roasting of comparatively small quantities of blende that may have been done in direct con-

[1] Annales des Mines de Belgique, VI, 1, 21.

nection with the distillation process, it may be considered that the actual distillation of 119,671 tons of spelter required the total consumption of coal reported, or 624,511÷119,671=5·2 tons per ton of metal produced. Assuming that the reduction material amounted to 40% of the weight of the ore smelted, or 121,500 tons, the consumption of heating coal for distillation, manufacture of retorts, power, etc., was 1·65 tons per ton of ore smelted. This is the general average of all kinds of furnaces, including direct fired and gas fired, the former in many cases of ancient designs and the latter with and without recuperation of heat. The ore smelted averaged 45·48% Zn and 4·35% Pb. Assuming that these averages apply to all the material smelted, including the between-products, its total zinc content was 138,144 tons and the average saving of metal as spelter 86·627%. Presumably this does not include the comparatively small additional quantity recovered and marketed as zinc dust.

Lynen states that the cost of distillation in Rhenish Prussia and Westphalia per metric ton of calcined ore is as follows: coal, 15s.; labor, 12s.; refractory material, 6s.; other supplies, repairs and renewals and general expense, 7s. Converting shillings into United States currency at the rate of 1s.=25c., the above figures are reduced to the following per short ton (2000 lb.): Coal, $3·40; labor, $2·72; refractory material, $1·37; other supplies, repairs and renewals and general expense, $1·59; total, $9·08.

In Upper Silesia the average cost of labor for the distillation and accessory process appears to be about $1·95@$2·375 per 2000 lb. of ore, not including the cost of roasting, varying according to the rate of wages.[1] At the comparatively high rate of wages that prevailed in 1900 the labor cost at one works was only $1·685 and at another $2·15 per 1000 kg. of ore, corresponding to $1·53 and $1·95 per 2000 lb. respectively. In both cases the distillation was done in gas-fired furnaces of large size. The average consumption of heating coal in Upper Silesia appears to be about 1·5 to 1·65 tons per ton of ore, a minimum of 0·8 having been reported in 1900 by one works which uses large, Siemens regenerative furnaces.

ITEMIZATION OF ACCOUNTS IN ZINC SMELTING.—The accounts of a zinc smelting works should be kept in such a manner as to show the actual cost of production in each step of the process. A convenient system of itemization, which is capable of considerably more or less elaboration and is easily conformed to, is as follows:

[1] According to the statistics of the Oberschlesischen Berg-und Hüttenmännischen Verein the average earnings of the men over 16 years of age employed in the zinc smelteries of that district were 811·32 marks in 1895; 829·53 in 1896; 833·87 in 1897; 895·70 in 1898; 922·83 in 1899; and 1005·63 in 1900. Estimating 300 working days per annum, these figures correspond to 60@80c. per day.

I. General expenses:
  a Interest, taxes and insurance.
  b Salaries.
  c Other expenses of administration.
II. Labor:
  a General.
  b Steam and power.
  c Yard labor and transportation of material.
  d Crushing and sampling ore.
  e Roasting.
  f Distillation.
  g Retort and condenser manufacture.
  h Repairs and renewals account.
III. Fuel:
  a Blende roasting furnaces.
  b Distillation furnace heating coal (including steam coal for blowing gas producers).
  c Reduction material (coal and coke).
  d Coal for steam and power (including heating of pottery).
  e Coal for heating chamotte kilns and tempering furnaces.
IV. Refractory material:
  a For retorts.
  b For condensers.
  c For repairing and rebuilding furnaces.
V. Other supplies, including all charges for repairs and renewals not elsewhere counted.

This system of accounts covers the entire cost of production at the works. There will be a further charge to be made against the product to cover selling expenses, either in the form of commissions or as salaries and expenses of sales agents. The cost of production is divided according to general expense, labor, fuel, refractory material, and other supplies, including repairs and renewals, which are the principal items that concern the commercial management of a zinc smelting business. It is important, however, for the technical management to be able to determine the cost according to the technical stages of the process, e.g., crushing and sampling, roasting, distillation, retorts and condensers, etc. The costs by such divisions are found readily by combining the cost of material under the respective captions with the corresponding cost for labor under that general caption. An elaborate system in the organization of a works is required, however, to permit an exact itemization of costs in that manner, which can only be done when miscellaneous supplies are segregated according to their use in the power plant, the roasting department, distillation department, crushing and sampling mill, etc. This involves the maintenance of a warehouse in charge of a capable storekeeper from which supplies will be debited to the various departments as called for. This is, however, an advisable system to adopt for more reasons than one. The close supervision of the stores which is made possible thereby tends to reduce waste of them; and charging them to the various departments where required promotes further

economy. All kinds of supplies purchased should be charged therefore to the storekeeper, who will be responsible for them and will in turn charge them to the several parts of the works where needed.

There is greater difficulty in keeping account of repairs and renewals, which it is nevertheless highly important to know. A portion of the supplies drawn from the warehouse, e.g., oil, waste, tools, etc., constitute articles of regular consumption, while another portion will be for the ordinary repairs and renewals, i.e., for maintenance of plant, and a third portion will be for the account of new construction. Moreover, the general labor of the works will be divided in an equally complex manner. In order to return those accounts segregated according to departments, the machinists, smiths, masons, carpenters, etc., should be in charge of a master mechanic (this, of course, contemplates a large works), who will keep a record of the employment of labor in his department and the material used as if he were conducting an independent business. In this way the aggregate cost of (1) repairs and renewals and (2) new construction will be known. There is a further difficulty in subdividing the cost of power between the various departments, e.g., the crushing and sampling mill, the pottery, the mechanical roasting furnaces and other parts of the works in which it is used, but which must be known if for example the cost of roasting is to be determined. Practically the only method by which this can be debited is to compute the cost of one horse power hour for the whole plant and charge each department according to the amount of power used.

The detail with which these accounts can be conveniently itemized differs naturally at various works, but as a general thing it may be assumed that the greater the detail in their preparation the better will be the commercial and technical results of the business, because of the better control over the work that is made possible. A well designed works should have arrangements for determining the weight and grade of the ore received, the weight and grade of the dry ore delivered to the roasting department, and the weight and grade of the desulphurized ore delivered by the latter to the distillation department. There should be provided also scales for determining the weight of the coal delivered to the various departments. The weight of the spelter produced will be known, of course. From these data the percentage of metal recovered from the ore and the losses in handling, roasting and distillation can be determined.

The difference between the value of products sold and the cost of production, repairs and renewals being charged in the latter, will represent the gross profit of the business. From the gross profit a charge must be made to cover amortization of the plant, because an industrial business to be

profitable must not only show a profit in operation, but also a profit large enough to return a reasonable percentage on the investment, besides redeeming the investment within a comparatively short period, say 10 years. If that period be adopted as that in which the works must return their first cost, an annual deduction of 10% from the gross profits must be made for the account of amortization. The remainder will be the net profit available for payment of dividends.

In reckoning an amortization period of 10 years, it is not contemplated that such a time will represent the life of the business, nor even that the plant will be worn out in that time, it being assumed that the repairs and renewals will be made with sufficient foresight and liberality to maintain the original plant at approximately its original efficiency, but a far-seeing management must be prepared to throw away a physically good plant, and replace it with something new, whenever the progress in industrial science has presented something better; and this policy can be followed properly only by having a reserve fund especially applicable to such a purpose. The deduction from the gross profits for amortization of plant will therefore be accumulated as a reserve fund, which may be drawn from for the construction of new plant, and if a proper deduction for such be made and if repairs and renewals be properly kept up it is unnecessary to charge new construction to operating expenses as is sometimes done. If no allowance for amortization be made, however, new construction ought to be charged directly to operating expenses. In that case profits are likely to exhibit great fluctuations from year to year, and there is no doubt that it is better business management to carry a proper amortization account and charge new construction against it.

Assuming that 10% per annum be a reasonable deduction from the gross profits for amortization of plant, a new industrial enterprise ought to be able to show a gross profit of 20 to 25% in order to make a proper return for the money invested, taking into consideration all the dangers of interruption to business by strikes, fire or destruction of property otherwise, risks in business management, danger of more powerful competition, etc.

XVI.

DESIGN AND CONSTRUCTION OF SMELTING WORKS.

The design and equipment of zinc smelting works differs so greatly that there is little use in attempting to select any typical example from practice. Most existing zinc smelting works both in Europe and America have been the result of a gradual development, and it is rare to find any works evolved in that manner in which the proper correlation of all parts has been observed. There are, however, various general principles governing the arrangement of a zinc smelting works in order to secure the best economical results which may be confidently laid down as a guide for the design of a new works. It will be understood, of course, that there may be as great a variety in the equipment of a zinc smelting works as in other kinds of smelting works, with the result that the total cost of the plant may range from $8 per ton of annual capacity (on the basis of blende assaying 60% Zn) to $16 per ton.[1] The equipment of a works is governed naturally by the economical conditions of the locality where they are to be situated. It is obviously unwise to build a works costing $16 per ton of capacity unless that high first cost can be shown to lead to a lower cost of smelting and higher recovery of metal as compared with a works costing only $8 per ton, sufficient to return a larger interest on the investment after deducting a proper amount for amortization of the capital. Accepting that a works can be built for as little as $8 per ton, the increase in cost over that figure will be due chiefly to the employment of a better grade of structural material, which will tend to reduce the repairs and renewal account in subsequent operation, and the introduction of means for handling mechanically the ore and coal and other material, for reducing the consumption of coal and for increasing the recovery of metal. Inasmuch as the chief factors in the cost of smelting zinc ore are in any case the cost of the coal and cost of the labor, and inasmuch as a large part of the cost of the latter is involved

[1] It is to be remarked that American zinc smelteries invariably include blende roasting departments, save in the few cases where calamine is the only ore treated. In Europe in many cases the roasting department is a separate installation.

in the handling of the large quantities of material which is necessary in zinc smelting, it is likely to be found that the gain in those items will more than offset the extra cost of plant up to a rather high figure.

LOCATION.—It is a well established principle in locating a zinc smelting works that in view of the large quantity of coal required in smelting zinc ore, which under the most favorable conditions is likely to be close to two tons of coal per ton of ore, it is cheaper to carry the ore to the coal than the coal to the ore. If the ore can be carried to coal in the direction of the final market for the products, rather than to coal in an opposite direction, so much the better. This subject involves, however, a consideration of the complicated questions of railway carriage and freight rates, and it is impossible to lay down general rules.

Topography of Ground.—In selecting a particular site for a works the points for chief consideration are a topography which will permit of the most harmonious arrangement of the various parts and the most economical storage of the huge quantity of waste products; i.e., coal ashes and retort residues, that will be produced. The question which used to trouble lead and copper metallurgists, and may do so still, as to whether a works should be located on a sloping or a level site, does not concern the zinc smelter. His plant will necessarily cover so large an area that it would be impossible to arrange the parts so that the ore would go from one step to the next by gravity, even if he wished it to do so. The ideal topography for a zinc smelting plant, in so far as the furnaces and buildings are concerned, is the smoothest and most nearly level that can be found. With a sloping and irregular surface the first cost of the works will be increased by the extra amount of grading that will be necessary and the outlay for retaining walls, of which none will be required if the site be level.

The question as to the method of building the distillation furnaces with respect to the general ground level has been discussed in Chapter X (q. v.). The tendency of not a few engineers is to build them entirely above ground (of course on a proper subterranean foundation), with an elevated operating platform, just as in the modern practice of installing open hearth steel furnaces. It may be urged that even if that be done it would be best to set the distillation furnaces on a lower terrace, so that the operating platform will be level with the ground on which the roasting furnaces are located and the ore can be trammed directly from the roasting furnaces to the distillation furnaces; or still better, locate on three terraces, the roasting furnaces on the first, cooling and storage floors on the second and distillation furnaces on the third. The objections to such an arrangement would be as follows: (1) the deposition of the ore from the roasting furnaces

on a cooling floor necessitates a rehandling of the entire quantity, which can be obviated by conveying mechanically the ore from the roasting furnaces to hopper-bottom bins, whence it may be drawn by gravity after cooling, and such an arrangement does not require a terrace; (2) in a modern works the charge will probably be mixed mechanically before going to the distillation furnaces, and since such a process implies a mechanical elevation of the material it may as well be delivered to the distillation

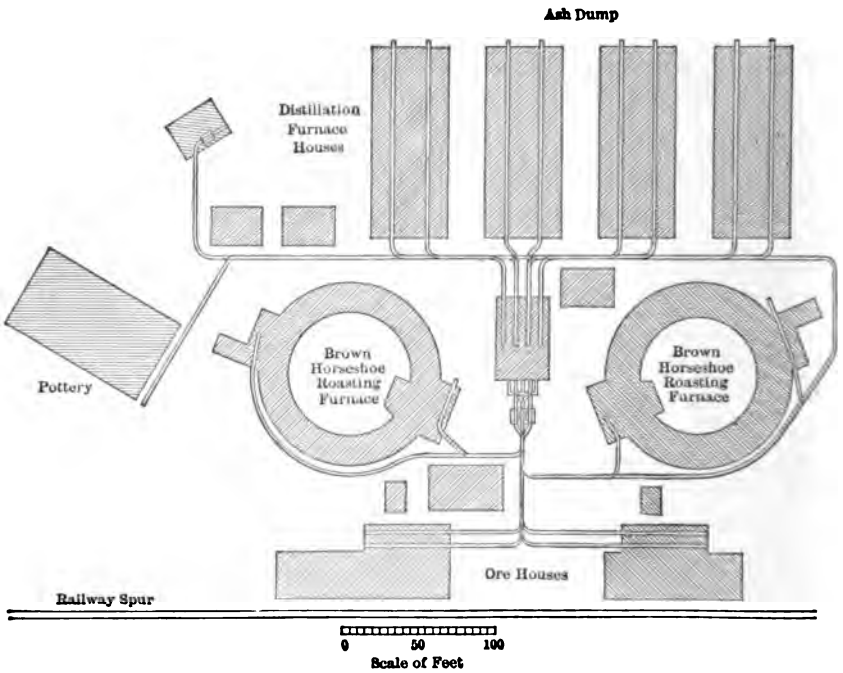


FIG. 394. PLAN OF WORKS OF EDGAR ZINC CO. AT CHERRYVALE, KAN.

At present this plant has one additional roasting furnace and two additional distillation furnace houses. Each of the distillation furnace houses comprises three furnaces of 200 retorts.

furnaces at any desired level. There would be therefore nothing to be gained from a three-terrace location and a good deal to be lost.

When the distillation furnaces are arranged in line, end to end, it is convenient to set the house at the beginning of a gentle declivity, which enables the ore to come in on the ground level on one side, and the ashes to go out upon the ground level on the other side. Numerous of the plants in Belgium are built on that plan, several of them being situated on the

bank of the River Meuse, which presents the desired slope. However, this requires a large excavation and a good deal of masonry, and the convenience is of doubtful advantage. It is feasible only with a single line of furnaces, extensions of which diminish the compactness of the plant, although it will be observed that in Belgium the furnaces are placed very near together and a good many of them are grouped in a house of very moderate dimensions.

Disposition of Ashes and Residuum.—Besides the arrangement of the furnaces and buildings of a zinc smeltery, a highly important matter in the consideration of the location of a works is the convenient disposition of the ashes and residues. The quantity of these that will be produced depends obviously upon the grade of the ore to be smelted, the quantity of fuel that has to be consumed, and the percentage of ash in the latter. These must be calculated for each particular case. It will be found, however, that even with the highest grade of ore, the smallest consumption of coal, and the lowest tenor of ash in the latter that can be expected, the quantity of ashes and residues produced will at least amount to 0·5 ton per ton of ore smelted, and may easily amount to one ton per ton of ore smelted. Retort residues weigh from 50 to 60 lb. per cu. ft., and coal ashes approximately the same, these weights being of the material in a loose form; under pressure the weight per cubic foot is of course increased to some extent. Assuming that one ton of ore makes only 0·5 ton of ash and that the latter weighs 50 lb. per cu. ft., the smelting of 10,000 tons of ore per annum will produce 5000 tons of residue, the storage of which will require 200,000 cu. ft. If the ashes be piled 10 ft. high, they will cover 20,000 sq. ft., or approximately half an acre. Ten years' storage would require therefore five acres; if the ashes were piled only half as high 10 acres would be required. These figures indicate the magnitude of the problem.

The ideal location of a zinc smeltery, with a view to disposition of the ashes, is a level area for the furnaces and buildings, abutting on a steep declivity, on the slope of which the ashes can be dumped. The distillation furnaces will be naturally aligned parallel with the slope, either with ends opposite or sides opposite, so that ashes will need to be trammed the minimum distance. The steeper the slope and the greater the fall, the less distance will the ashes have to be conveyed and the less the area that will be covered by their accumulation. If such a topography be unavailable, and if we go to the other extreme of convenience, namely, a level or nearly level prairie, such as constitutes the location of many works in Kansas, the problem of ash disposition becomes most serious. At such works, if they be small and uneconomically operated, it is the common custom to accumulate the ashes in unsightly mounds around the works, up the slopes

of which they must be hauled in carts. If a large works designed with a view to economical operation were confronted by such an unfavorable topography, the solution of the difficulty would be obviously in a system of elevating mechanically the ashes, of the nature of those which are employed for the disposition of slag at some lead and copper smelting works.

The retort residues are used to a considerable extent in the United States for railroad ballast, being a very good material for that purpose. In the Western States, where stone and gravel are comparatively expensive, the railway companies are frequently glad to remove the residues, if the latter are deposited in such a way that they can be cheaply loaded on the cars, and sometimes may be willing to pay a small price for them.

STORAGE OF ORE, COAL AND MATERIAL.—In large smelting works there must be provided ample capacity for the storage of ore, coal and other supplies, so that the operation of the works may be independent of temporary difficulties at the mines or on the railways. As a general rule, the farther the works are distant from the mines, the greater the stock of ore that must be carried, not only because of the longer period that the ore will be in transit, but also because of the more difficulties that may happen in the longer line of supply. There will be less difference in the stock of coal, because it is presupposed that in any case the works will be situated near a coal supply. The stock of fire clay is of less consequence, partly because the quantity required is proportionately small and partly because the supply of finished retorts and condensers that must be kept in stock is necessarily so large that a temporary cutting off of the raw material would not be likely to cause trouble. The question of retorts and condensers has been discussed in a previous chapter relating to the manufacture of those articles.

The ore being chiefly in the form of fine concentrates will come to the works in box cars, which must be unloaded by hand, there being no efficient mechanical method of discharging that type of car. A sample of the ore will be taken undoubtedly during the unloading of the car, that being a convenient time, and it being desirable to know the composition of every lot of ore as soon as it is received in the works. As to the storage of the sampled ore opinions may differ. It is entirely a question of cost. The method which is most expensive in installation but cheapest in subsequent operation is to run the railway cars up an incline to a trestle behind a line of bins with sloping bottoms, from which bins the ore can be drawn into a small tram-car, or a mechanical conveyor, without the necessity of the laborious rehandling, which will be necessary if the ore be dumped into bins on the ground level. Another way is to keep the railway cars on the

general ground level and push each car to be unloaded opposite a chute leading to the boot of a belt elevator; the ore being dumped into the chute is elevated to any height desired and discharged into a small receiving hopper, whence it is drawn into a tram-car and conveyed to the final storage bin; the plants of the Lanyon Zinc Co. at Iola, Kan., are arranged in this manner, which gives command of a large storage area, but unless the bins are made self-discharging the ore has to be rehandled, of course, by shoveling. In small works, where the quantity of ore carried in stock is not large, the general method is to unload the cars directly into a line of bins on the ground level. Inasmuch as the floor of the ordinary box car is 4 ft. above the rails and concentrated blende is heavy material (it is quite safe to reckon Joplin concentrates at 125 lb. per cu. ft. or 16 cu. ft.=2000 lb.) storage for a considerable quantity of ore is obtained in that manner without excessively long leads in wheeling or having to wheel the barrows up grade.[1]

The cars having been weighed upon receipt at the works and samples of the ore having been taken as they were unloaded, the metallurgist knows not only the weight but also the composition of the ore in each pocket or bin and is able to draw therefrom as desired in order to make any desirable mixture.

The storage of coal is a good deal more difficult problem than the storage of ore, inasmuch as the quantity required is greater and instead of following a single line through the works, it must be distributed to numerous places. The method of storage will be so much dependent upon the method of distribution throughout the works that no general principle can be laid down. This is a matter which should be left to the consulting engineer. It may be said that the coal will probably be received in hopper-bottomed cars, and arrangements should be made to take the coal from them. Hopper-bottomed cars for the transport of coal are now in general use in the eastern part of the United States, and it is only a matter of a few years before they will undoubtedly be used exclusively in all parts of the United States, since the old type of cars is rapidly being destroyed by making up into trains with heavier cars and it is being replaced by the more modern form. The coal storage problem is one from which the natural gas smelters are fortunately free to a large extent.

CRUSHING MILL.—The choice and arrangement of the machinery for crushing blende and calamine, preparatory to roasting and distillation, do

[1] A bin 16 ft. wide loaded with ore to the height of 4 ft. will hold four tons per linear foot; to store 1000 tons of ore there must be therefore 250 linear feet of bins of the above dimensions of width and height. A bin 8 ft. deep holds two tons per linear foot.

not differ from those adapted to the moderately fine crushing of other ores. It is not so essential to avoid the production of an excessive proportion of fines as it is in some other kinds of crushing, but nevertheless it is desirable to produce as uniform a product as possible, since therewith both roasting and distillation can be more closely regulated. For the first crushing of lump ore there is no better type of machine than the Blake crushers which are made by reliable manufacturers; for the fine crushing, rolls are undoubtedly the most efficient. The crushing of blende is a comparatively simple matter, since it breaks easily and its high specific gravity increases the capacity in tons of any given machine as compared with the capacity of the same machine for crushing a quartzose ore. For this reason zinc smelters are apt to install crushers and rolls of a cheap and inferior grade, but although they may stand up to the work it is poor economy to employ crushing machinery of poor design and construction. On the other hand there are some grades of these machines commanding high prices because of the special reputation of their manufacturers, acquired when most of the other makes were very inferior, which are seldom worth their extra cost. High grade and highly efficient rolls are now available at a cost midway between the extremes.

Rolls.—For the kind of crushing usually required in connection with a zinc smeltery belted rolls driven at a speed of 100 r. p. m., or so as to have a peripheral velocity of from 600 to 1000 ft. per minute, may be advantageously used,[1] their capacity being greatly increased at that speed as compared with the speed of 40 or 50 r. p. m. at which geared rolls are commonly driven. The shells may be either of chilled iron or of hammered steel. The former will suffice for the comparatively easy work of crushing zinc ore, but the latter will be the more satisfactory as to preservation of an even surface and have the advantage that they can easily be turned true when worn; on the whole they are likely to be the cheaper. In fine crushing the importance of feeding the rolls evenly, in order to get the best work and longest life from them, cannot be overestimated. If the rolls are fed directly by hand this will depend upon the attention of the man engaged in that work; if they are fed from a chute it is advisable to interpose a mechanical feeding device between the end of the chute and the hopper of the rolls in order to insure an even distribution of the ore. In installing a crushing plant machinery of ample capacity should be provided; the most economical work will be attained when the capacity materially exceeds the duty that such machinery is called upon to perform. In estimating capac-

[1] Rolls of 27 in. diameter at 100 r. p. m. have a peripheral speed of 709 ft.; those of 36 in. diameter at the same speed make 942 ft., while 22 in. make 576 ft. per min.

ity, however, it is proper to remember that zinc blende is not only brittle and easily crushed, but also that bulk for bulk it is considerably heavier than quartz, wherefore the output in tons of a given machine will be proportionately greater.

If nothing but washed ore is received at the smeltery, as is the case at many works in Kansas which smelt nothing but Joplin concentrates, jaw crushers are of course unnecessary. With such material it is advisable to screen it before passing it to the rolls. Joplin concentrates will all pass a 0·5 in. round hole and from 50 to 60% by weight will pass a standard eight-mesh wire cloth, which is about the size required by the roasting furnaces. The taking out of that material, which is already fine enough, relieves the rolls of so much work and increases the capacity of the mill.

Dryers.—It is the common practice among the zinc smelters of Kansas to dry the ore before crushing it, although the percentage of moisture in the general run of the concentrated blende received from Joplin and vicinity is by no means high. By a preliminary drying, the capacity of the rolls and especially of the screens is greatly increased. If the ore is to be dried, an efficient type of apparatus should be selected. Probably there is nothing better than the well constructed revolving cylinder dryer, unless it be a modern zigzag tower dryer; if either be used proper precautions should be taken to avoid loss of fine ore in the form of dust carried off by the chimney draught; this implies the installation of a dust-settling chamber between the cylinder, or tower, and the chimney.

The simplest form of dryer is a series of cast iron plates placed over a flue so that they will be heated from below by the hot gases from the fireplace, each plate having a flange at its sides through which adjoining plates can be bolted together if desired, by which method fine ore is prevented from sifting through the joints between the plates. The use of such a dryer, however, necessitates hand labor. The zigzag, or gravity, dryer is a rational and efficient device, but a belt elevator must generally be installed in connection with it. This form of dryer has been highly developed in the Edison tower, which is built in the form of a shaft, 4×4 ft. in area and 24 ft. or more in height. These dryers, which are employed at the magnetic separating works of the New Jersey Zinc Co. at Franklin Furnace, N. J., occupy small floor space at the expense of height, but in mill buildings the latter is generally cheaper than the former. Another type of dryer which deserves attention is a series of troughs heated from below, through which the ore is made to travel by means of endless screws, the whole design being in fact an adaptation of the ordinary screw conveyor. Although the wear and tear of such apparatus is rather great, it is at its

best when handling fine ore, and if the various parts are properly designed the cost of repairs and renewals should not be excessive. The movement of the ore through a trough conveyor can be effected, however, by traveling rakes in the same manner as in a mechanical roasting furnace, or by the push-plates of the ordinary scraper conveyor. The drawbacks to making dryers out of trough conveyors are the great expense for the large heating surface that must be provided and the considerable consumption of power in their operation.[1]

The dryers commonly employed in Kansas are platforms of iron plate supported by brick walls and heated from below by a direct fire or by the waste heat from roasting furnaces; the older works, at Pittsburg and vicin-

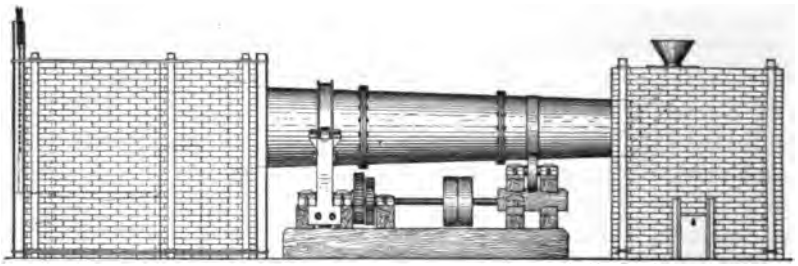


FIG. 395. ROTARY DRYER.
18 ft. × 44 in.

ity, used commonly an unlined cylinder of boiler plate, revolved by suitable gearing, with a fireplace at one end and an iron chimney at the other. These cylindrical dryers were much cruder than those which are used in the gold and silver mills of the Rocky Mountains (as shown in Fig. 395), the construction being less substantial and the loss of heat by radiation from the iron cylinder, destitute of brick lining, being greater; while no arrangement was made for collecting the dust, which was of course lost through the chimney. A new form of dryer, used at Iola, is a circular iron plate (about 14 ft. in diameter) heated from below, upon which the ore is fed at the center, whence it is worked spirally toward a peripheral discharge by a set of revolving plows.

The loss of ore as dust is likely to be no unimportant matter in drying fine ore in a revolving cylinder, and the more the capacity of the latter be increased by faster driving, or harder firing (increasing the velocity of the combustion gases through the cylinder) the greater will be the loss in dust. Hence a cylindrical dryer should be of large size, partly for the

[1] The increasing use of magnetic separators, which must be provided with perfectly dry ore, has caused the question of dryers to become rather an important one.

sake of exposing a greater surface of ore to the hot gas, thus permitting a slower speed, and partly to reduce the velocity of the gas passing through the cylinder. Both the Rothwell dryer, which consists of a cylinder divided into four quadrants by longitudinal diaphragms, and the Argall dryer,[1] which consists of four small cylinders united by surrounding rings, are logical improvements over the simple cylindrical dryer.

Screens.—With respect to screens there is probably nothing more efficient than the old revolving trommels; at least none of the shaking-riddles, gyratory screens and various patented devices has proved to be better as yet. Revolving screens may be either cylindrical or prismoidal (usually hexagonal); with sides parallel or tapering; if the former the shaft has to be set on an incline with special boxes for the journals; if the latter the shaft may be horizontal, the ordinary boxes being employed. In either case the inclination of the lower side of the screen should be about 1 in. to the foot, sloping toward the discharge. The ordinary screen, 3 ft. in diameter, should be driven at about 20 r. p. m. The driving gear should be arranged so that when necessary to stop one screen it can be thrown out of gear without stopping the whole mill. It is the best practice to arrange the screening surface in segments, whatever be the type of the trommel, so that when a break appears it is necessary to replace only the particular segment in which it occurs with another which has been previously made ready. The covering may be either of ordinary heavy steel wire cloth or of steel plate punched with slots or square holes.

Wire cloth screens, having the larger percentage of open space, give a greater capacity than steel plate screens with holes of the same diameter, but the steel plate is the more durable, and on that account is generally to be preferred. The holes in the wire cloth being approximately square and those of the steel plate being circular the latter give the more uniform product. Steel plate screens with square holes and rectangular slots are sometimes used, but those with circular holes are the preferable and also the most common. In steel plate screens with circular holes the latter are punched in rows making angles of 90° with each other, or else the holes are staggered. The latter form gives the greater percentage of open space. The limit to the number of holes is reached when the strength of the plate is too much reduced; consequently the holes may be punched nearer together in a thick plate than in a thin one. If the holes be staggered and one half diameter apart there will be 40·3% open space; if in rectangular

[1] The Argall dryer is similar in design and construction to the Argall cylindrical roasting furnace. The dryer was indeed used originally for roasting gold ore preparatory to lixiviation with potassium cyanide solution at Cripple Creek and Florence, Colo.

rows, 34·9%; if the holes are one diameter apart the percentages are 22·6 and 19·6 respectively.

In referring to wire cloth for screening purposes the custom is to state the number of meshes per linear inch, but inasmuch as wire of various diameter is employed in making cloth of a certain number of meshes per linear inch, the character of the cloth cannot be determined unless the number of the wire from which it is woven be specified. There used to be a good deal of confusion with respect to wire cloth, owing to the different wire gauges employed by different manufacturers. This difficulty has, however, been happily obviated, the Wire Cloth Manufacturers' Association having adopted on March 7, 1899, the Washburn & Moen, or Worcester, gauge as the standard for all iron or steel wire cloth and the old English gauge as the standard for all brass or copper wire cloth. The mesh in wire cloth is always the number of openings in a linear inch measuring from center to center of wire. A linear inch will comprise, therefore, the same number of apertures and wires. Multiplying the diameter of a single wire in decimals of an inch by the number of wires in a linear inch, subtracting the product from unity and dividing the remainder by the number of meshes per linear inch, will give the diameter of each aperture in decimals of an inch. The dimensions of standard, heavy mining wire cloth are as follows:

Mesh	Wire No.	Diam. of wire inches	Diam. of aperture inches	Equivalent in millimeters
1	3	·2437	·7563	19·81
¾	4	·2253	·5247	13·33
⅔	5	·2070	·4180	10·62
2	8	·1620	·3380	8·59
2½	9	·1483	·2517	6·39
3	10	·1350	·1983	5·04
3½	11	·1205	·1652	4·20
4	12	·1055	·1445	3·67
4½	13	·0915	·1307	3·32
5	13	·0915	·1085	2·76
6	14	·0800	·0867	2·20
7	15	·0720	·0709	1·80
8	16	·0625	·0625	1·59
9	17	·0540	·0571	1·45
10	18	·0475	·0525	1·33
12	19	·0410	·0423	1·07
14	20	·0348	·0366	·93
16	22	·0286	·0339	·86
18	23	·0258	·0298	·76
20	24	·0230	·0270	·69
22	25	·0204	·0251	·64
24	26	·0181	·0236	·60
30	28	·0162	·0171	·43
40	31	·0132	·0118	·30
50	34	·0104	·0096	·24
60	36	·0090	·0077	·20
64	37	·0085	·0071	·18
70	38	·0080	·0063	·16
74	39	·0075	·0060	·15
80	40	·0070	·0055	·14

A roll of wire cloth is always understood to be 100 linear feet. Any width can be obtained to order. Prices are quoted per square foot.

Elevators.—The elevators that must be used are generally of the band and bucket type. The bands will be of rubber belting, except in the case of the elevator taking the discharge from the dryer if the latter be so hot that a rubber belt would be softened, when link belts or chains passing over sprockets must be used; it is seldom to be anticipated, however, that the ore will be delivered so hot as to necessitate that type of elevator. Belt and bucket elevators may be either vertical or inclined. The vertical elevator usually adapts itself the better to the construction of the building, and permits often a simpler arrangement of the machinery; on the other hand the inclined elevator naturally maintains a regular tension of the belt on the head wheel, from which the motion is imparted, while the vertical elevator must have an adjustable boot in order to regulate the tension. In either case the boots should be arranged so that the bottom may be dropped for the purpose of cleaning out accumulations of ore there.

The feed of ore to the elevator may be through a chute leading into the housing about 18 in. above the center line of the boot wheel, the idea being that the ore will be shot directly into the buckets, or the feed may be into the bottom of the boot, so that the buckets will simply scoop it up as they pass around the boot wheel. Mill men differ as to which is the better method, but with fine, dry ore there need be no fear of tearing off buckets in scooping up ore which is fed into the bottom of the boot. A good discharge at the head of the elevator may be obtained with either the vertical or the inclined type if the chute be properly set. In installing belt elevators, it is particularly essential to give them an ample capacity. The buckets should never be reckoned as running more than 33⅓% full. They are generally spaced about 18 in. apart. The belt should be 2 in. wider than the buckets. It is seldom advisable to make the head wheel less than 30 in. in diameter even with light belts. The boot wheel is commonly made of smaller diameter than the head wheel. The best speed of belt is about 300 ft. per minute. Belt elevators are frequently referred to as a necessary evil, but if they are substantially constructed, and especially if they are designed with ample capacity, they need not be expected to cause more trouble than any other piece of machinery.

Blende Roasting Plant.—In arranging the roasting furnaces the chief essential is to provide ample room around them. If they be reverberatories, raked from each side, the distance between a pair of furnaces should be at least 16 ft. in order that the men may be able to handle their tools and work comfortably. It is far better, however, to set them 20 ft. apart, which

involves only the small extra cost of the building; assuming that the building will be 80 ft. wide, the furnaces being set at right angles to its longitudinal axis, and is to cost 50c. per square foot the difference in first cost between 20 ft. and 16 ft. spaces is only $4\times80=320$ sq. ft., or $160 per furnace, which is very little in view of the extra convenience which is gained. With mechanical roasting furnaces it is unnecessary to provide so much space between them, but even with them it is best to have as much as 16 ft. If the furnace be a single hearth reverberatory, the throat end may be built in line with the side wall of the building, the longitudinal tie rods being tightened up from the outside of the latter if the siding be of corrugated iron, of which pieces may be easily removed; or the end of the furnace may set inside of the building at a sufficient space from the wall to give access to the buckstaves, damper, etc., from the inside; the latter is the preferable arrangement. Between the fire box and the other side of the building there should be at least 12 ft. and preferably 16 ft. in order to leave room for the convenient handling of coal, roasted ore, etc.

The small blocks of shelf burners formerly used at Pittsburg, Kan., were almost always installed in an independent house, and the larger blocks of the same type now used at Iola are sometimes housed in similar manner, but more commonly they are grouped in one large building, either parallel or at right angles to the axis thereof. The two-hearth reverberatories, used for blende roasting, are generally set end to end, on the center line of the building; this is necessary because of the position of the fireplace, i.e., in the middle of the block. In the case of muffle furnaces their location with respect to the Glover towers, so as to obtain the shortest and most direct lines for the gas flues, is a governing consideration, it being important to deliver the gas as hot as possible, within reasonable limits, to the tower in order to secure the most effective operation of the latter.[1]

Whatever be the type of furnace the ore should be charged through a hole in the roof from a hopper above the furnace, the hopper being of sufficient capacity to hold at least one charge. If desirable, in order to avoid tramming at night, to have ready the required supply of ore for the night shift the ore hopper may be made double, each part containing a charge. The ore hoppers may be suspended from the lower chords of the roof trusses, or they may be supported from independent columns capped by steel I-beams or channel bars spanning the furnace, the latter arrangement being generally the better one. Either arrangement is preferable to having the ore hoppers rest upon the masonry of the furnace.

In a systematic arrangement the roasting furnaces will be naturally

[1] Vide pp. 48, 177, 178 and 179.

aligned, so as to permit the installation in the simplest manner of mechanical devices for conveying ore, coal, etc. In this connection it must be remembered, however, that nearly every mechanical conveyor requires attention and if the work to be done is within the capacity of one man per day it is seldom economical to install a long and expensive conveyor. In most cases the cheapest method of transporting the ore to the roasting furnaces will be found a simple elevated tramway, extending through the roasting house, passing over the ore hoppers above the furnaces, into which the ore will be discharged from the bottom-dump cars that will be employed. If the furnaces be direct fired the coal may be brought to the furnaces on a similar tramway and discharged into hoppers over the fire boxes, from which it may be dropped into the latter through a hole in the roof. Unless the coal be of very inferior quality, however, the quantity of it that is required is relatively small, and it is not likely that it will be advisable to go to the expense of that installation. When the fire box is at the end of the furnace the simplest method of delivering coal is to bring in a spur track of the railway behind the fire boxes of the row and unload the coal from the cars in which received directly into a bin back of the furnace from which it will be drawn as required; the bin behind each furnace should have capacity to hold at least a carload of coal.

The building covering the roasting furnaces will have naturally a pitched roof. This may be arranged with its ridge at right angles to the individual furnaces, in which case the roof trusses will necessarily have a comparatively wide span, and will be correspondingly expensive, with the advantage, however, that there will be no posts in the house; or the roof may be designed with a series of valleys and ridges, making a kind of M-roof in which the ridge plates will be parallel with the long axis of the furnaces. In the latter case there will be necessarily some posts in the building; such posts if properly placed close to the furnaces and midway between the working doors need not interfere, however, with the use of the long tools, and by trussing the wall plates the number of posts can be reduced. Whatever way the roof be designed, the ridges should be capped by large monitors with louvers to insure good ventilation. For the same reason the roof should not be of less than 1:3 pitch. It is generally advisable to make the building of such height that there will be at least 7 ft. above the tramway over the furnaces up to the tie beams of the roof trusses, but if because of the character of the furnaces this would give an excessive height to the building, the roof trusses can be designed so that the tramway will come through one of their panels, over the tie beams. The building should have doors of ample dimensions, say 7×7 ft., conveniently placed and a sufficient number

of windows in the sides to give good light. It is useless to glaze the windows of a furnace building; they should be closed with shutters, either hinged or pivoted.

MIXING ORE AND REDUCTION MATERIAL.—The ore discharged hot from the roasting furnaces must be permitted to cool before anything further can be done with it. The simplest method is to wheel it from the furnaces to a brick cooling floor, which must of course be roofed over. When cold it is weighed out and mixed with a proportion of crushed calamine, or any other kind of ore that is to be used in the charge, and with the percentage of reduction material specified by the metallurgist. The reduction material, which will probably be a mixture of lean coal and coke, will be prepared most conveniently in a mill adjoining the mixing house. The neces-

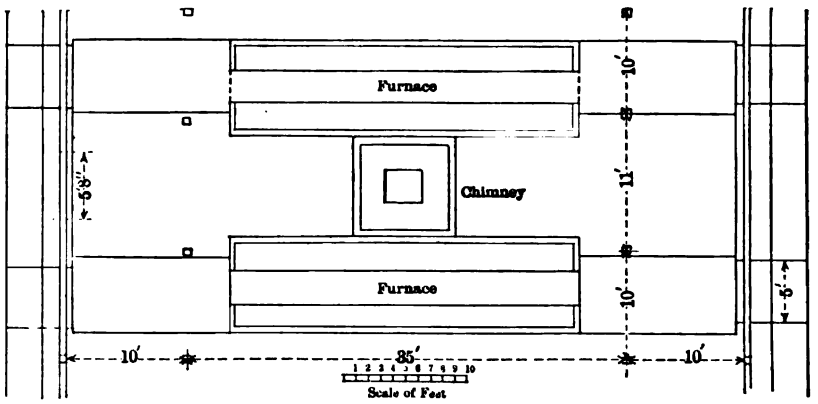


FIG. 396. ARRANGEMENT OF DISTILLATION FURNACES AT PULASKI, VA.

Plan.

sity for permitting the hot ore to cool before mixing it with the reduction material involves of course a rehandling of the entire quantity, the expense of which is reduced to the minimum if the ore be accumulated in elevated storage pockets, whence it can be drawn by gravity.

The ore and reduction material weighed out in the proper proportions may be incorporated either by hand or by machinery. If by hand the work is done generally on the floor in front of each distillation furnace where the components of the charge are dumped separately. It is more systematic to do this work mechanically, however, and a better mixture is effected thereby, which is desirable. The apparatus employed for that purpose has been described in a previous chapter. The prepared charge is most conveniently deposited in a car, of which there should be one or more for each furnace.

The floors of the distillation furnace houses can then be kept clear, the charge being conveyed thither only when it is wanted, i.e., the first thing in the morning.

DISTILLATION DEPARTMENT.—The various ways in which the distillation furnaces may be arranged as to level have been referred to in a previous chapter. The location of a number of furnaces with respect to each other will depend obviously a great deal upon the type of furnace that is to be employed, and it is difficult to prescribe general rules. It may be said,

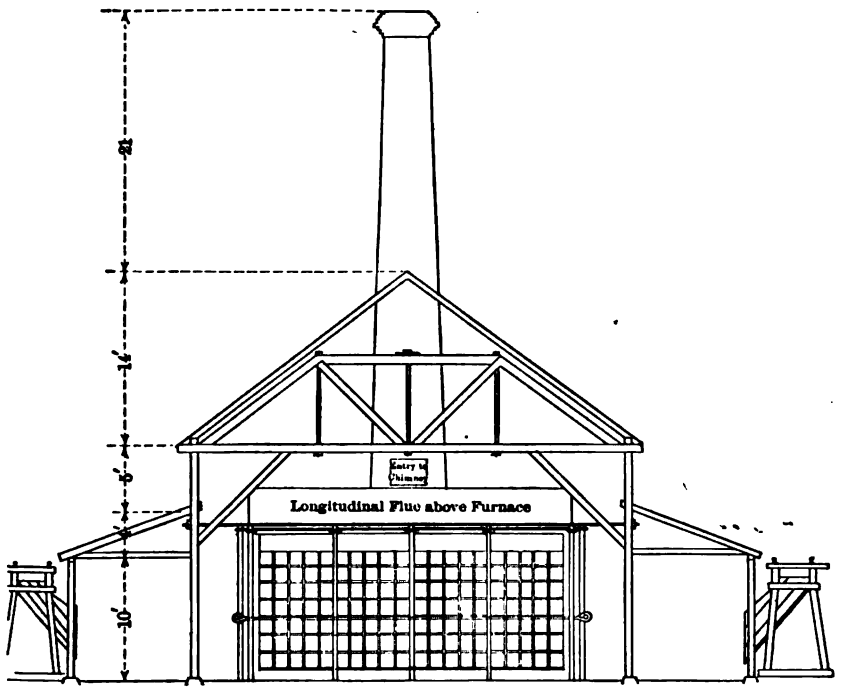


FIG. 397. ARRANGEMENT OF DISTILLATION FURNACES AT PULASKI, VA.
Elevation.

however, that as a general thing there ought to be a space of at least 18 ft. in front of each furnace. With direct-fired furnaces in which the fire doors are placed in the end walls there should be a space of about 14 ft. between the latter and the corresponding side wall of the building, in order to afford room for convenient stoking and the necessary supply of coal. A direct-fired Belgian furnace with 224 retorts (112 per side) will require therefore a house about 50 ft. square. A series of furnaces may be set end to end

in one long house, which is the common practice in Belgium, but with that arrangement (which is, of course, the most economical of room) the delivery of the large quantity of coal that is required cannot be made so conveniently as when the furnaces are set with their fronts parallel. In making the latter arrangement it is inadvisable to attempt to cover more than one furnace with the same roof, since two opposite furnaces must be set at such great distance apart, in order to avoid subjecting the men to an unbearable heat, that a building of extravagant dimensions is required. There is no doubt that the best arrangement for a series of direct-fired Belgian furnaces is with their façades parallel, their end walls aligned and each contained in its own house, which need be only a comparatively inexpensive shed, following the general practice in Kansas. This arrangement makes it easy to loop an elevated cable tramway around the entire row of furnaces and dump the coal required by each furnace directly in front of the fire doors at each end. The houses need not be set more than 25 ft. apart, but 40 ft. is preferable, inasmuch as it diminishes the danger of the destruction of the entire department by fire in case a single house burns. In the case of gas fired furnaces in connection with which the producers are placed outside of the building, the furnaces are most conveniently set end to end in one or more long houses, the gas producers being capable of various arrangements which will permit the easy delivery of coal to them.

POTTERY.—The pottery should be placed conveniently to the tempering furnaces, which in their turn must be placed conveniently to the distillation furnaces. Because of the considerable amount of power and steam that is required in it, however, it should also be placed near the steam and power plant, but in locating the pottery above all things attention must be given to safeguarding it from fire, the pottery being the key to the entire plant, of which the operation would necessarily be suspended for many months in the event of its destruction.

STEAM AND POWER PLANT.—In a small smeltery wherein the consumption of steam and power will not be very large the plant for their development must be placed near the points where they are to be used, i.e., the crushing mill, mixing mill and pottery; or at least those departments should be situated so that they can be reached by a simple rope transmission. Power can also be conveyed, of course, for comparatively short distances in the form of steam if the pipes be properly protected against loss of heat by radiation, and where it is necessary to employ independent engines as for driving mechanical roasting furnaces, the power will very likely be transmitted by that method. It is desirable, however, to avoid the necessity of having independent engines in the pottery and crushing mill. In large

plants, wherein a large amount of power may be required, it is of course a simple matter to establish a central station with high class boilers and engines and directly connected generators to convert the mechanical power into electrical power, which may be transmitted elastically. The amount of power required in small zinc smelting plants is too insignificant to make advisable the employment of compound engines, but even in the case of small engines, there is a good deal of choice, and the kind should be selected which will develop a brake-horse power upon the smallest number of pounds of steam. High class boilers are generally advisable, but they are not intrinsically more economical of coal than an ordinary tubular boiler.

Railway Spurs and Tramways.—In the erection of a new plant no permanent building ought to be commenced until the arrangement of the whole plant has been definitely decided upon and laid out on paper. That having been done, it will be possible to arrange the railway spurs and works tramway system in the manner which will be most economical. It should be aimed to bring in the ore and coal, which represent the greatest bulk of material that will come by railway, in such a manner that the cars can be unloaded directly into the bins from which the material is to be used. The simpler the arrangement of the works in that respect, the less will be the cost of a mechanical plant for the conveying of the material about the works. For the latter purpose there is no system which is so cheap in first cost and also in operation, when large quantities of material have to be handled, as the cable railway. In addition to cable railways, etc., the works will require a more or less elaborate system of tramways for the handling of comparatively small quantities of material in various directions. Above all things, all the tramways of a works should have a uniform gauge. The latter may be conveniently made 20 in., measuring between the heads of the rails. With that gauge curves should be given a radius of 12 ft. to insure easy running of the cars. The loads which the latter are likely to carry will seldom necessitate the use of rails heavier than 12 lb. to the yard.

Buildings.—Furnace buildings are most cheaply erected with timber frames sheathed with 1 in. boards, the roof boards being covered with some good kind of ready roofing. With lumber at $20 per M the cost of rough siding per square is about $2·50, which is only increased a little if battens be put on in addition. The roof boards and putting on will come to about the same figure per square, and the prepared covering will cost about $2@$2·50 per square additional. If corrugated iron be used it should be of No. 22 or No. 24 gauge, not lighter than the latter. The former weighs 1·38 lb. per sq. ft., and when laid with ordinary laps the weight will be 160 lb. per square. Its cost in place on the roof will generally be $6@$7

per square. There may be a slight saving in the purlins, but so slight as to be insignificant.

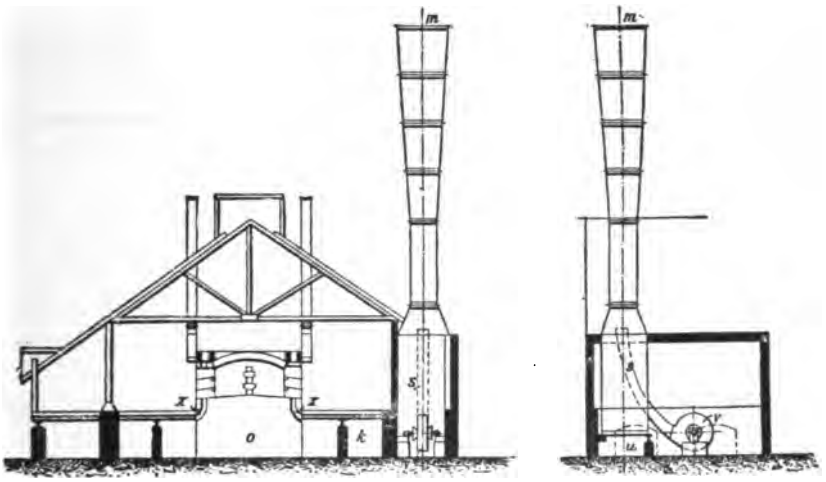
It is useless to employ galvanized iron, because the sulphurous fumes will quickly destroy the zinc coating of the latter. Ordinary corrugated iron should be well painted, if used, since its life depends chiefly upon the thoroughness with which it is coated. Silica graphite paint is excellent for that purpose; it costs about 20c. per 100 sq. ft. more than iron oxide paint. It is frequently stated that where sulphurous gases are very abundant, and especially in moist climates, sheet iron should not be used at all, but there are many furnace buildings in which it has been and is used with satisfactory results when properly cared for. Buildings of corrugated iron on a framework of structural steel may be employed, but they are usually more expensive than the timber frame. In framing a building with timber it is advisable to avoid mortises and tenons in favor of simple butt joints, securing the beams, posts, etc., by bolts and nuts, lag screws, drift bolts, spikes, treenails, dowels, etc. Such joints are not only cheaper than those prescribed in fine carpentry but also are stronger.

Good foundations are essential to all buildings, furnaces and machinery; but much money is often squandered in making them unnecessarily solid. It is unnecessary to go deeper than to undisturbed gravel or sand, which point is as often reached 3 ft. from the surface as more; in all parts of the United States where zinc smelting is practiced a depth of 3 ft. below the surface is beneath the frost line. A cheap, expeditious and satisfactory method of setting the foundations for a furnace building is simply to dig holes and set a good sound log directly under where each post of the building will come, filling around the logs with rammed stone and then sawing off their upper ends perfectly true for the sills of the building to rest upon them. For furnaces and machinery a large block of concrete, made of one part cement and two to three parts of clean, sharp sand, thoroughly mixed while dry and after being made into a stiff paste added to six times its bulk of broken stone, makes an excellent foundation. Good uniform sand is an excellent support even for very heavy loads, such as tall chimneys, providing that the weight be distributed so that there will be not more than two tons per square foot. Good, firm gravel will safely stand a load of four tons per square foot.

The floor of the distillation furnace house should in all cases be paved. It is also desirable to pave the floors of the roasting furnace houses. In the United States brick is commonly employed for such pavements. Care should be taken to put down a good foundation for it, retort residues being a suitable material; otherwise it is likely to become uneven and loose, and

will be an eyesore and an inconvenience, in a very short time. If the working floor be an upper story of the furnace house, and is of timber construction, a brick pavement should be laid on the plank.

In Europe, the furnace buildings are commonly constructed with view to greater permanency than in the United States; also timber is not so cheap there. Brick walls, pierced by the necessary openings for access, light and ventilation are the common practice. In the more modern buildings the roofs are supported by steel trusses. The working floors are paved and supported on brick arches if there be a lower floor, which is becoming a common arrangement.



FIGS. 398 AND 399. SYSTEM OF FURNACE HOUSE VENTILATION AT OVERPELT, BELGIUM.

Fig. 398: End elevation. Fig. 399: Side elevation of fan and chimney.

SANITARY ARRANGEMENTS.—The labor on the distillation furnaces is so severe that whatever arrangements conducive to the health and comfort of the men can be made will tend toward improvement in their work. It is essential to provide the men with a supply of good drinking water, which they want ice cold. In the hot weather of summer ice forms a considerable item in the miscellaneous supplies entered in the cost of smelting. Good ventilation of the furnace houses is the chief thing to look out for. An opening in the roof, directly over each furnace and for at least the length thereof, is advisable. The ventilation of the furnace house is promoted also by giving the roof a sharp pitch. Nothing less than 1:3 should be permitted and 1:2, i.e. 45°, is preferable. In Belgium the furnace houses

usually have roofs of about 40° pitch (vide Chapter XVII). A new arrangement, installed in the works at Budel, Holland, and at Overpelt, Belgium, has been described by F. Kiessling.[1] The system at Overpelt is shown in Figs. 398 to 400. The furnaces are of the Siemens type, installed two in a house, with working floors 2·5 m. above ground level, as described in Chapter XVII. The residues drop through the chutes *xx* into the spacious cellar. Formerly these chutes were closed by gates so that during the maneuver there would be no up-draught through them, but in spite of that the development of smoke and dust was so great as to give trouble. The closed chutes gave trouble, moreover, because the residues of easily

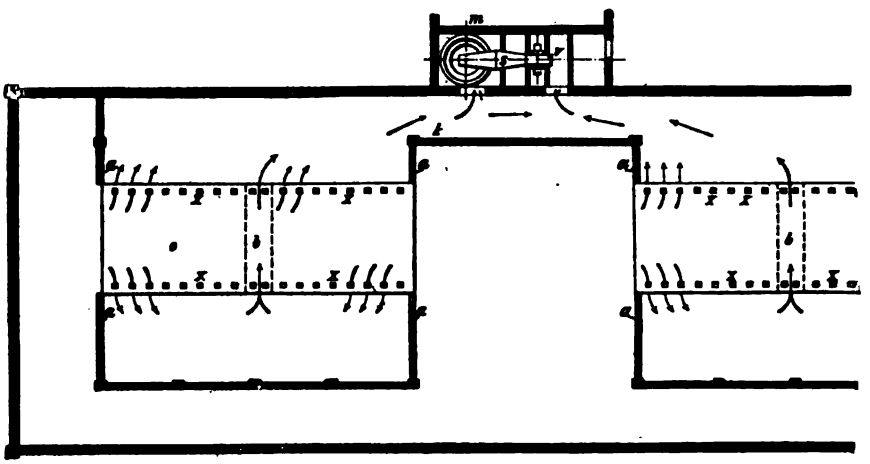


FIG. 400. SYSTEM OF FURNACE HOUSE VENTILATION AT OVERPELT, BELGIUM.
Plan.

fusible charges would stick in them on cooling, so that they could only be removed with difficulty. In the present arrangement the chutes are left open and the residues fall directly into the cellar. From the cellar, which is enclosed by the exterior walls and the iron doors *aa*, the air is exhausted by the fan *r* and discharged through the tube *s* into the iron chimney *m*, which is 2·5 m. in diameter at top and bottom, 20 m. high, and of the shape shown in the engravings. The fan is 1·8 m. in diameter, and driven at 550 to 600 r. p. m., exhausts 50 cu. m. of air per second. Since the installation of this system the furnace house has been free from smoke and dust

[1] Berg-u. Hüttenm. Ztg., LXI, xxxviii, 478, Sept. 19, 1902.

and the draught removes the hot air from the fronts of the furnace, making it more comfortable for the men, and thereby shortening the time required for the maneuver and increasing the time for distillation, which is important.

Fumes of zinc oxide appear to be quite harmless to human beings, but when the ore subjected to distillation contains lead there is danger of poisoning by the fumes of that metal. In Europe the arrangement of a zinc smeltery with respect to sanitary conditions is prescribed by law. In this connection the new German law of February 6, 1900, is of special interest, as showing the precautions which the experience in that country has pointed out. Some portions of it are summarized in the following paragraphs.

The buildings in which roasting and distillation furnaces are installed should be spacious and well ventilated. Floors should be level and smooth, permitting the easy removal of dust, preferably after dampening it. The walls should be of a character that will prevent the accumulation of dust, and if of such kind that they cannot be washed they should be whitewashed with lime at least once a year. Accumulations of dust on the tops of furnaces should be removed at frequent intervals. An abundant supply of water should be provided in each house, and the floor should be wet down at least once a day. The workmen should be provided with good drinking water without having to go out into the open air to obtain it. Proper means should be provided to prevent the escape of dust from ore crushing machinery. The ore must be moistened before charging into the retorts, except in the case of Silesian muffles, when it need be moistened only if it is of such character that its dust is particularly dangerous to health.

The gases escaping from distillation furnaces should be conducted to the exterior of the building by a strong draught as soon as possible after their issuance from the condensers. Residues from the retorts may not be drawn out into the furnace house, but must be dropped into suitable pockets below the floor. If blue powder is sifted it must be done in such a way as to prevent the escape of dust.

Suitable lavatories and places for eating, kept free from dust and dirt, must be provided outside of the workrooms; workmen are not permitted to eat their luncheons in the latter.

There must be at least a space of 6 m. (20 ft.) between the front of a distillation furnace and the opposite wall of the building. When two furnaces are set with opposite faces (long axes parallel) the distance between them must be at least 10 m. (33 ft.). Galleries below the floors of distillation furnace houses must be at least 3·5 m. high, and must be light and well ventilated.

XVII.

EXAMPLES FROM PRACTICE.

In the chapters which have preceded this, it has been aimed to present the results of experience in the metallurgy of zinc according to the separate steps in the smelting process. In addition thereto, it is useful to segregate descriptions of the practice at various works. Consequently there has been collected in this chapter some notes of the results in distillation and accessory processes, which have been obtained at smelteries in Europe and the United States, it being designed so far as possible to avoid duplication of what has been written in the previous chapters.

Austria.

Cilli.—At the zinc smeltery at Cilli, Austria, a Siemens furnace with 136 muffles is employed. The muffles, which are 0·20 m. wide and 0·42 m. high, are made of one third raw clay and two thirds chamotte. The charge per muffle amounts to 35 kg. As reduction material a mixture of cinder and brown coal is employed; as fuel a mixture of stone coal and lignite. Each furnace has two gas producers. One charge is distilled per 24 hours. The retort residues assay 4 to 5% Zn.[1]

Belgium.

The following descriptions of the zinc smelteries of Belgium are taken except where other authorities are cited, from a paper by Ad. Firket, inspector general of mines, in the *Annales des Mines de Belgique,* 1901, VI, i and ii. His investigation and report, which referred especially to the sanitary condition of the works, embodied much valuable data as to the design and arrangement of the various plants, which have been abstracted herein. The conditions described are what they were at the end of 1898. The following statement of the production of the various works in 1898, in metric tons, shows their relative importance:

[1] Berg-u. Hüttenm. Ztg., LXXXIV, 31.

Name of Company	Name of Works	Place	Production
Soc. Anon. de la Vieille Montagne	Valentin-Cocq	Hollogne-aux-Pierres	24,397
" " " " " "	Angleur	Angleur	16,270
" " " " " "	Flône	Hermalle-sous-Huy	9,110
Soc. Anon. des établissements Dumont et frères	Sart-de-Seilles	Seilles	12.146
Soc. Anon. de la Nouvelle Montagne	Engis	Engis	10,591
Soc. Anon. Austro-Belge	Corphalie	Antheit	10,179
Soc. Anon. métallurgique de Prayon	Prayon	Forêt	8,736
Soc. Anon. métallurgique de Boom	Boom	Boom	7,450
L. de Laminne	La Croix-Rouge	Antheit	7,013
Compagnie d'Escombrera-Bleyberg	Bleyberg	Montzen	5,406
Soc. Anon. des métaux d'Overpelt	Overpelt	Overpelt	4,760
Soc. Anon. des fonderies de Biache St. Vaast	Ougrée	Ougrée	3,613

ANGLEUR.—The double furnaces of the Société Anonyme de la Vieille Montagne at Angleur, contain 200 retorts, 100 on each side. Two furnaces are built in a block, which thus has 400 retorts. They are fired with gas and the air of secondary combustion is preheated. The retorts receive an average charge of 30 kg. of ore mixed with 12 kg. of reduction coal, and produce in 24 hours from 12 to 15 kg. of zinc, besides a certain quantity of blue powder. In the production of 1 kg. of zinc from 3·5 to 4·5 kg. of reduction and heating coal are consumed. The loss of zinc amounts to 15%. The retorts last according to their quality and their position in the furnace from 15 days to four months. A furnace holds out for two to three years.[1]

The furnaces employed at Angleur have been described in Chapter X. The latest constructions have only 320 retorts per massive, arranged in four rows. At the end of 1898 the works comprised 17 massives and another one was being built. The retorts are elliptical, 0·16×0·23 m., and 1·4 m. in length. The condensers are formed with a bulge for the zinc to collect in. Conical prolongs are employed. Six massives are grouped in a house, the three houses being arranged on three parallel lines. Each house is about 100 m. long. The distance between massives is 2 to 2·8 m. One house is 15·4 m. wide; another, 14·75 m.; and the newest, 17·6 m. The height of the walls is about 6 m. The roofs pitch 40°, 36° and 39° respectively. The floors of the subterranean galleries for removal of the residues are about 2·75 m. below the working floor. The ores smelted at Angleur are low in lead and the residues are not further worked.

Antheit.—The Usine de la Croix-Rouge (L. de Laminne) has 34 direct fired Dor furnaces with step grates, each comprising five rows of 10 retorts, or a total of 1700 retorts. Thirty-two of the furnaces are grouped in fours, forming eight massives; the ninth massive comprises only two furnaces. These are installed in two houses, a large and a small. The former

[1] Bulletin de la Société de l'Industrie Minérale, 1888, p. 505

contains seven of the large massives. It is 17 m. in width inside, and 100 m. in length; it is divided into two unequal sections by a transverse wall. The height is 11 m. at the middle and 3·65 m. at the sides. The roof, which is supported by wooden trusses, has a pitch of 39°; throughout its entire length there is an opening, 2·2 m. in width, over the furnaces. The façades of the latter are on the average about 6 m. from the walls of the building, and about 6 m. intervenes between the ends of the adjacent massives. The small house is of about the same design as the large one, but it is only 27 m. long. Both houses are built on the bank of the River Meuse, the working floor being 2·6 m. above the ground level on the lower side. Originally the ashes were removed through sub-galleries, 1·5 m. wide and 2 m. high, which extended transversely under each massive, but the more part of the plant has been reconstructed so that there is a clear room, 2·3 m. in height, beneath the working floor, the latter being supported by iron columns and brick arches. The retort residues are now dropped into hoppers, whence they can be drawn by gravity into cars. The ores distilled at Antheit are high in lead. The residues assay 9 to 12% Pb and about 165 g. Ag per 1000 kg. They are concentrated to a product assaying about 31·9% Pb and 410 g. Ag per 1000 kg. At Ampsin there is a blende roasting plant with eight three-hearth reverberatory furnaces, with three working doors per hearth.

Boom.—The plant of the Société Anonyme Metallurgique de Boom comprises nine massives of 240 retorts, each massive consisting of four furnaces of 6×10 retorts. The furnaces are direct fired, Liège type, with step grates. There are four houses. One contains three massives, separated by spaces of 4·25 m. The others have each two massives, separated by a space of 5·5 m. The houses are 15·15 m. wide inside, leaving a clear room of 5·45 m. in front of the furnaces, on each side. From the working floor to the roof the height is 4·3 m. at the sides and 10·6 m. in the middle, the pitch of the roof being about 35·5° The ash galleries are on the ground level, the working floor being 3·45 m. above the latter. The working floor is extended by platforms 2·5 m. wide, on each side of the house, for the convenient handling of ore, spelter, etc. The working floor is laid on arches, which afford a height of 3·25 m. to the lower story. The furnaces (massives) are 8·58 m. long and 4·25 m. wide. The residues drop into pockets through slots, 0·4 m. wide, in the floor along the fronts of the furnaces. In the roof, above each massive, there is an opening 5·5 m. long and 3 m. wide, boxed up to the height of 3·5 m., for the purpose of ventilation. The retort residues, which contain about 9% Pb, are concentrated by screening and jigging to 30 or 35% Pb.

Corphalie.—The single furnaces employed at this plant, which have been described in a previous chapter, are grouped in pairs, end to end; each furnace having 7×10 retorts, the massive comprises 140. The massives, 21 in number, are arranged, about 2 m. apart, in a straight line, the houses having a total length of 276 m. The latter are 15 m. in width inside, and 10·5 and 3·5 m. in height at the middle and sides respectively, the roof having a pitch of 43·5°. In the sides of the houses there are numerous openings and in the roof one of 2·2 m. width for the whole length of the latter. The line upon which the faces of the furnace front coincides with the middle line of the building, leaving a clear space of 7·5 m. before the furnaces and 4·8 m. behind. The working floor, which is 4·55 m. above the ground level, is a pavement laid on brick arches filled in between steel beams, leaving a clear height of 4·3 m. to the lower story. In the walls of the latter there are openings 2 m. in width and 3·3 m. in height, affording good light and ventilation. The residues from the retorts drop through an opening 0·3 m. wide, in front of each furnace, into commodious pockets, whence they are removed, after cooling, in cars.

The ores treated at Corphalie are high in iron and lead. The retort residues contain on the average about 8·5% Pb and 14 to 15% Fe, the latter partly in the state of metallic iron and magnetic oxide. They are first screened, the undersize passing to a magnetic separator for removal of the iron and the oversize to a crusher. The product of the latter, together with the tailings from the magnetic separator, is jigged, and yields a concentrate with 40 to 45% Pb. The blue powder collected from the prolongs is remelted in a Montefiore furnace, comprising 12 boots. Blende is roasted at Corphalie in a plant of 12 double hearth reverberatory furnaces, with three doors per hearth, which are combined back to back in six massives.

Engis.—At the end of 1898 these works comprised 13 furnaces of the Nouvelle Montagne type,[1] containing an aggregate of 1274 retorts, 12 of these furnaces being united end to end so as to form six massives and the remaining one being single; two massives of Dor direct fired furnaces, containing 440 retorts (four furnaces being united in a massive and each furnace having five or six rows of 10 retorts); and four Siemens furnaces, each containing 240 large retorts. Plans were in hand at that time for the replacement of a number of the old direct fired furnaces by two additional Siemens furnaces.

The direct fired furnaces are contained in a house 13·9 m. in width inside, and 140 m. in length. Two transverse walls divide the house into three sections, the middle one, which is 20 m. in length, serving for the

[1] Vide p. 431.

storage of coal. The walls of the two furnace rooms are pierced by numerous large openings for light and ventilation. The distance from the fronts of the furnaces to the opposite walls of the building is 5·1 m.; the minimum distance between the furnace massives is 3·75 m. The furnace rooms are 12·1 m. in height in the middle, the pitch of the roof being 41°. There is a longitudinal opening in the roof, over the furnaces, 2·1 m. in width. The house sets along the bank of the River Meuse. There is a longitudinal sub-gallery, 1·4 m. in width and 2·15 m. in height, extending under the line of furnaces, and numerous transverse galleries, the latter debouching at the ground level on the down hill side. The retort residues drop through openings, 0·2 m. in width, in front of the furnaces.

The Siemens furnaces are contained in two houses, two in each. One of them is 13·75 m. in width and 57 m. in length, its height being 9·9 m. at the middle and 3·5 m. at the sides. The façades of the furnaces are distant 4·4 m. from the walls, and the two furnaces set 13 m. apart. The other house is 16·9 m. in width and 44 m. in length. The increased width affords a clear room of 6 m. on each long side of the furnaces; the latter are set only 4·4 m. apart, longitudinally. The house is 11·5 m. in height in the middle; the roof pitches 39°. The working floors are 2·4 m. above the ground level.

Ores high in lead are treated at Engis, the residues containing 8 to 12% Pb. By screening, crushing and jigging they are concentrated to a product assaying 30 to 40% Pb and 400 to 500 g. Ag per 1000 kg.

Flône.—The furnaces employed at this plant of the Société Anonyme de la Vieille Montagne are of the same type as those used at Valentin-Cocq, but the retorts are somewhat smaller. Two furnaces are grouped end to end as a massive, each furnace having 108 retorts, or 216 per massive. There are 12 massives, which are contained in a house 190 m. in length. Its older portion, 155 m. in length, is 12 m. in width inside, affording a clear room of only 3·4 m. in front of the furnaces. It is 10·85 m. high in the middle and 4·85 m. at the sides, the roof having a pitch of 45°. Above each massive there is an opening 2·5 m. in width. These used to be covered by monitors, but it was found that the latter interfered with the dispersion of the fumes and they were consequently removed; without them the ventilation is excellent. The newer extension of the house is 18·8 m. in width, and 14·3 m. and 6·5 m. in height at the middle and sides respectively. The distance between the massives is 3·5 m. and the clear room in front of them is 6·75 m. on each side. In this extension the roof has a pitch of 39°. The faces of the furnaces are provided with counterweighted doors, corresponding in width to that of the niches and in height to that of two

rows of retorts. These are used when the retorts are being discharged. The residues drop into pockets below the working floor, whence they are removed in cars on a monorail system. At the four corners of each furnace there is a sheet iron chimney, rising above the roof of the house, to ventilate the pockets in which the residues are collected.

Besides the distillation furnaces, there are at Flône nine reverberatory roasting furnaces. These are now attended by two men per furnace, working eight-hour shifts, instead of three men working 12-hour shifts, as formerly. The number of men per 24 hours and the output per furnace remain the same.

Forêt.—The Société Anonyme Métallurgique de Prayon has two houses, each containing six Siemens massives of 240 retorts.[1] The older of the two is 150 m. long and 18·8 m. wide inside, the fronts of the furnaces being 6·9 m. from the longitudinal walls of the building, which are pierced by numerous and large openings. Interposed between the massives, which are arranged in line, there are three furnaces for tempering retorts. The distance between the adjacent distillation furnaces is 6 to 6·5 m., but when a tempering furnace intervenes the distance is 11 to 12 m. The newer house is 125 m. long and 18·4 m. wide inside, the clear space in front of the massives being 6·7 m. and the distance between two massives, end to end, 5 m. In this installation the tempering furnaces are outside of the main building. The roofs of the two houses pitch 45°. There is an opening 2 m. wide throughout their length, which in conjunction with the sharp pitch insure good ventilation. The roofs are of iron. The height from the working floor to the roof is 13 m. at the center. The working floors are 2·7 m. above the ground level.

The Prayon furnaces have been described elsewhere in this book. Cylindrical prolongs are used on the condensers. The blende that is treated is roasted in a neighboring sulphuric acid works.

Montzen.—The new plant, which is now the only one in operation, consists of a house 142 m. in length and 17·8 m. in width inside. The height is 6 m. at the sides and 12·5 m. in the center; the roof (which is supported by iron trusses) has a pitch of 34·5°. The house contains three Liège massives of 336 retorts (four furnaces, each 7×12); three Dor massives of 240 retorts (four furnaces, each 6×10); and two Loiseau massives of 288 retorts (two furnaces of 144 per massive). These furnaces are arranged in line. The house is ventilated by a monitor 3·5 m. in width, extending the whole length of the roof. The ash galleries are 1·5 m. wide and 2·4 m. high, except in those places where the transverse flues of the Loiseau fur-

[1] An older plant of six direct fired blocks has lately been dismantled.

naces pass, in which they are necessarily lower.[1] The level of the galleries is 3·4 m below the working floor of the house. The ground level corresponds with the floor level on one side of the house and with the gallery level on the other side. The retort residues drop into pockets, whence they are drawn in tram-cars, the galleries being laid with rails.[2]

The residues at Montzen, containing about 5% Pb, are jigged so as to afford a first class product assaying about 24% Pb and 316 g. Ag per 1000 kg., and a second class product assaying about 13% Pb and 160 g. Ag.

The works at Montzen are operated by the Compagnie d'Escombrera-Bleyberg. The roasting plant and the old distillation plant are idle.

Ougrée.—The works of the Société Anonyme de Biache St. Vaast comprises four Loiseau furnaces with 144 retorts each, one Loiseau massive of 288 retorts, and one Dor massive of 200 retorts. These are contained in a house 17·3 m. in width inside and about 67 m. in length. It is an old construction, being only 2·3 m. high at the side walls, and 8·8 m. in the middle, and the ventilation is poor. The distance between the furnaces, end to end, is about 4 m.; the space in front of the furnaces is 6·7 m. on one side and 6·2 m. on the other. The subterranean galleries are 1·6 m. wide and 2 m. in height to the crown of the arch, the latter being a semicircle. There is one gallery extending longitudinally under the line of furnaces and four transverse galleries.

The ore treated is exclusively calamine, high in iron and low in lead, containing about 45% Zn on the average. The residuum is screened and passed over magnetic separators to recover the large quantity of metallic iron and Fe_3O_4 which it contains. The tailings therefrom are jigged for recovery of a further portion of iron. The washed coal which is obtained by the jigging is given away.

Overpelt.—This plant has six Siemens furnaces of 216 retorts each, contained in three houses (two in each). The houses are 15·4 m. wide inside and about 50 m. long. The roof, supported by iron trusses, is 4 m. above the working floor at the sides and 10·5 m. at the center, the pitch being 38°. Along the ridge there is an opening about 20 m. long, which is boxed up to a height of 0·5 m. above the top of the roof, with converging sides, so that the opening is 2·5 m. wide at the bottom and 2·3 m. at the top. The working floors of the houses are 2·5 m. above the ground level. The faces of the furnaces are 5·2 m. from the longitudinal walls. The distance between the two massives in each house is 10 m., a tempering furnace inter-

[1] Vide description of the Loiseau furnace, p. 446.

[2] At many of the Belgian works, even some of the important ones, the residues have to be shoveled.

vening. On the lower floor (ground level) there is an eating-room and baths for the men.[1]

The general design of the furnaces at Overpelt was described in Chapter X. The condensers fit in the upper part of the retorts (the axes of which are 0·17×0·30 m.), the lower part being closed during the distillation by means of a tile, which is removed when the residues have to be raked out, the condensers remaining in place during the maneuver. Cylindrical prolongs are used on the condensers. In raking out the residues the niches are closed in front by means of doors, except of course the one that is being worked through, and the ashes drop through holes in the iron shelves into pockets below the floor level, whence after cooling they are transferred to cars. Originally, at this plant the residues were dropped directly into covered cars, but that method had to be abandoned because in cooling they agglomerated to such an extent that it was difficult to empty the cars and the latter were rapidly destroyed. The ores treated at Overpelt are high in lead, the residues assaying about 14% Pb. They are concentrated by jigging to about 38% Pb.

Seilles.—This plant comprises 14 massives of direct fired Liège furnaces, each of 240 retorts (6×10 per furnace and four furnaces per massive). The massives are 8·75 m. long and 4·7 m. wide. They are arranged in line, about 6 m. apart, in three houses.[2] The latter are 17·2 m. in width inside, and 11·25 and 3·5 m. in height at the middle and sides respectively, the roof having a pitch of 40°. The fronts of the furnaces are 6·25 m. from the longitudinal walls of the building. The sub-floor is 3 m. below the working floor. On one side of the house the working floor corresponds with the ground level; on the other side the sub-floor is at the ground level, the latter sloping. The system of ventilation is very good. In the whole length of the roof there is an opening 2·6 m. in width, above the furnaces. Hanging from the roof, the entire length of the building, there is a sheet iron curtain on each side of the line of furnaces, which descends to the level of the top of the latter. These curtains, which are 5 m. high, converge slightly toward the roof; at the top they are 7·4 m. apart; at the bottom, 7·9. Their lower edges are 1·6 m. beyond the faces of the furnace, and 0·85 m. beyond the line of the orifices of the prolongs. This arrangement produces an active draught. The prolongs used on the condensers are conical. The residues from the retorts are dropped through openings 0·25 m. wide, along the fronts of the furnaces, into hoppers, which are twice as wide as the openings in order to avoid trouble from agglomeration. The ores

[1] A more recent arrangement is described on p. 630.

[2] The most recent of these installations was made in 1898.

treated are high in lead, the residues containing 8·1 to 9·5% Pb. Blendes are roasted in Eichhorn-Liebig furnaces in a sulphuric acid works, which is connected with the smeltery.

VALENTIN-COCQ.—At the works of the Société Anonyme de la Vieille Montagne at Valentin-Cocq direct fired Silesian furnaces of 24 muffles (1·3 m. long, 0·55 m. high and 0·22 m. wide) and furnaces with 32 muffles (1·4 m. long, 0·6 m. high and 0·22 m. wide) were originally employed. The 24-retort furnaces were charged with 580 kg. of calamine and 150 kg. of reduction coal, and required a consumption of 1808 kg. of heating coal per 24 hours. The breakage of muffles per 24 hours was 0·41 and of condensers two. The 32-retort furnaces were charged with 920 kg. of calamine and 230 kg. of reduction coal, requiring a consumption of 2143 kg. of heating coal per 24 hours. The breakage of muffles per 24 hours was 0·4 and of condensers 2·4.[1]

The furnaces now employed at Valentin-Cocq are of the Rhenish type, with three rows of 18 retorts per side, 108 per furnace and 216 per massive, two furnaces being combined end to end. These furnaces, together with the old Silesian furnaces referred to in the preceding paragraph, were described and illustrated in Chapter X. The plant, which in point of magnitude of spelter production is the largest single plant in the world, comprises 34 massives of two furnaces and one single furnace, the total number of retorts being 7452.

This great number of furnaces is arranged in line in a series of houses, of which the total length is about 640 m., the houses themselves being set on parallel lines, the distance between them being 7·5 to 12·5 m. In the more recent constructions the width inside is 17·5 m., affording a clear space of 6 m. opposite each façade of the furnaces. The latter are 15 m. in length (per massive) and a space of 3 m. intervenes between each pair of massives. The retort residues and ashes from the grates are removed through longitudinal and transverse galleries, 1·5 m. in width and 2·4 to 2·5 in height, which extend under the furnaces. Rails are laid in the galleries for tramcars. The galleries are entirely subterranean; for purpose of ventilation they open into uncovered trenches, outside of the furnace houses, or communicate with shafts, which are covered by gratings. The ashes and residues are piled in a heap on the other side of the river from that on which the furnaces are situated, to which they are elevated and conveyed over a bridge.

This smeltery was established by the Soc. Anon. de fonderies de zinc de Valentin-Cocq in 1846 and was purchased by the Vieille Montagne in 1853.

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 173.

GERMANY.

BERNHARDIHÜTTE, UPPER SILESIA.—This works, which is one of the most recent constructions in Upper Silesia, has eight Siemens regenerative furnaces of 80 muffles each, a total of 640 muffles. The Siemens furnaces were originally introduced into Upper Silesia by the metallurgists of G. von Giesche's Erben and are now used to the exclusion of other types in the works of that company. The furnaces at the Bernhardihütte are noteworthy for their great size, containing a larger number of muffles than any other furnaces in Upper Silesia. In 1900 the Bernhardihütte smelted 15,058 metric tons of calamine, 7898 of roasted blende and 46 of between-products, a total of 23,002 tons. The consumption of cinder was 8719 tons, or 37·9%. Heating coal of various kinds, chiefly of the better grades, amounted to 27,665 tons, of which 2622 was for calcining calamine, 238 for drying muffles, 520 for tempering muffles and 24,285 for heating the distillation furnaces. The total consumption of coal for distillation, exclusive of cinder, but inclusive of what was used for preparing muffles, was 1·09 ton per ton of ore; for the distillation furnaces alone it was only 1·05 ton per ton of ore. The yield of spelter was 4108 tons, or 17·86% of the weight of the ore. The consumption of coal for distillation was approximately six tons per ton of spelter produced. The consumption of fire clay was 1576 tons=0·07 per ton of ore. Of muffles, there were used 5112; estimating an average working time of 345 days, this number corresponds to a breakage of 14·8 per day, or 2·3%. On the same basis the average charge per muffle was 104 kg. and per furnace 8320 kg. The cost for labor was 150,520 marks for men (149), 1279 for boys (4), and 10,961 for women (25), a total of 162,760, or 7·08 marks ($1·685) per ton of ore =39·62 marks ($9·43) per ton of spelter.

BIRKENGANGHÜTTE, RHENISH PRUSSIA.—At the Birkenganghütte, near Stolberg, Siemens furnaces with 108 retorts, arranged in three rows of 18 each, i.e., 54 per side, are used (1895). The ore distilled consists chiefly of roasted blende assaying 52 to 53% Zn, which is mixed with 37 to 38% of lean coal as reduction material. The charge for a 108-retort furnace amounts to 4·9 metric tons of ore, which is distilled in 24 hours, 1·2 ton of coal being burned in the gas producer per ton of ore. The furnace crew per 24 hours numbers 10 men, who work 12-hour shifts. The retorts are made of one third raw clay and two thirds chamotte; they last 40 days on the average.[1] At the Birkenganghütte there are also furnaces installed with recuperators for heating the secondary air only. These furnaces have

[1] Schnabel, Handbuch der Metallhüttenkunde, II, 178.

108 retorts (54 per side) arranged in three rows of 18 each, like the Siemens furnaces. They are charged with the same quantity of ore and give practically the same results as the Siemens furnaces.

CARLSHÜTTE, UPPER SILESIA.—This works has ten Siemens furnaces of 32 muffles each. In 1900 they smelted 7745 tons of ore (of which 1963 was roasted blende and the remainder calamine). The consumption of cinder was 52% of the weight of the ore and of fuel (coal slack) for all purposes, except blende roasting, 1·7 tons per ton of ore, or approximately 10·7 tons per ton of spelter produced (1229 tons).

CLARAHÜTTE, UPPER SILESIA.—This works has twelve direct fired furnaces of 24 muffles each, a total of 288. In 1900, there were smelted 3867 tons of calamine, 2813 of roasted blende and 673 of between-products, a total of 7353 tons, which yielded 1291 tons of spelter, or 17·56%. The consumption of cinder was 57·5% of the weight of the ore and of slack coal for heating the furnaces, etc., 1·5 ton per ton of ore, i.e., approximately nine tons per ton of spelter produced. The breakage of muffles was 2603, or approximately 2·6% per day. The average charge of ore per muffle was about 75 kg. and per furnace 1800 kg. Labor cost 9·72 marks ($2·31) per ton of ore=55·35 marks ($12·17) per ton of spelter.

DORTMUND, WESTPHALIA.—The plant at this place in 1894 comprised 10 massives, each of 240 retorts. The ore treated was almost exclusively blende. The average grade of the roasted ore was 50% Zn. The furnace charge was 8000 kg. of ore and 2400 kg. of lean coal as reduction material, which yielded 3600 kg. of spelter assaying 98·5% Zn. The loss in distillation was 10 to 12%. The roasted ore contained 0·4 to 1·1% S, according to the nature of its gangue. Retorts lasted for 60 charges on the average, the consumption being approximately 0·5 per ton of ore smelted. They were made of two parts burned clay and one part of raw clay, both from Andenne, Belgium, to which 10% of coke dust (by volume) was added, the mixture being thoroughly effected by shoveling and then by pugging with 10% of water in a horizontal mill, 2·5 m. long, in which the 27 knives made 7 r. p. m. and ejected the sausage through a 0·25 m. tube. The sausage, cut up into sections, was pounded into ballots, 0·4×0·4 m., by means of a 50 kg. drop hammer, from which the retorts were molded, at the rate of 70 per day, by a Dor hydraulic press. The retorts were 1·25×0·37×0·22 m., outside measurements, the walls being 30 mm. thick. They dried one month in the pressroom, five months in the next room, and six months in the hotroom. The crew of each distillation furnace (massive) comprised two stokers, each working a 12-hour shift, four brigadiers and four smelters (grand-manœuvres), besides eight helpers (petit-manœuvres) during the

time of discharging and recharging the retorts. The latter was begun at 5 a. m. and finished at 11 a. m. Spelter was drawn only once per 24 hours, namely, at 4 a. m. The consumption of coal was 1·5 ton per ton of ore.[1] The roasting furnaces in use at Dortmund, and the distillation furnaces, have been described in previous chapters of this book.

GLADBACH, WESTPHALIA.—At the works of the Gesellschaft Berzelius, at Gladbach, furnaces containing 208 retorts, arranged in three rows, 104 per side, are used. One charge is distilled per 24 hours, the furnace crew for 24 hours consisting of 12 men. The ore distilled is roasted blende, assaying 52% Zn, mixed with cinder as reduction material, the charge per retort amounting to 25 kg. of ore with 8 kg. of cinder and per furnace 5200 kg. ore with 1664 kg. cinder, which yields 2400 kg. of spelter, the loss in distillation being 10 to 13%. In the production of 1 kg. of spelter 3·9 kg. of heating and reduction of coal are consumed. Retorts last from 45 to 60 days.[2] The consumption of heating coal according to the foregoing data would be about 1·5 ton per ton of ore.

GUIDOTTOHÜTTE, CHROPACZOW, UPPER SILESIA.—The erection of these works, in 1887, occupied seven months and cost upward of 1,000,000 marks ($238,000).[3] They comprised 16 gas fired distillation furnaces of 32 muffles each, a total of 512 muffles, having capacity for smelting about 19,000 tons of calcined ore per annum; a plant of Hasenclever furnaces, capable of roasting about 14,000 tons per annum; and a sulphurous acid department, from which 857 tons of liquid anhydride were turned out in 1891.

HOHENLOHEHÜTTE, UPPER SILESIA.—The distillation furnaces in use at these works contain 64 muffles of 56 cm. height and 15 cm. width; those adjacent to the gas shafts are 1·46 m. long; the others are 1·66 m. long. The charge per 32 muffles consists of 850 kg. of roasted blende assaying 42 to 45% Zn, 250 kg. of lump and washed calamine assaying 26 to 32% Zn, and 1050 kg. of calamine slimes assaying 14 to 16% Zn, the total weight of the charge amounting to 2300 kg., from which 490 kg. of spelter is obtained for a consumption of 3·35 tons of run of mine coal in the gas producers.[4] This does not include the reduction coal.

The ore is roasted at the Hohenlohehütte in double hearth reverberatory furnaces (Freibergeröfen), of which the hearths are about 7 m. long. The sulphurous gases are conducted to towers 6 m. high where they rise through a spray of lime water, or milk of lime, calcium sulphite being formed. The sulphurous anhydride which escapes combination is dispersed by a

[1] Revue Universelle des Mines, 1894, XXV, 38.

[2] Schnabel, Handbuch der Metallhüttenkunde, vol. II.

[3] Th. Dahlblom, Berg-u. Hüttenm. Ztg., Dec. 11, 1891, p. 449.

[4] Dammer, Handbuch der Chemischer Technologie, II, 673.

chimney 100 m. high. The calcium sulphite is oxidized, by exposure to the atmosphere, to calcium sulphate which is sold as a fertilizer. Before roasting the ore is pulverized to 2 mm. size. The distillation furnaces are heated by Boetius generators, two per furnace, operated with an under-grate blast. The furnaces are claimed to give an average campaign of four to five years. The muffles are made of three parts raw clay, which is a mixture of Silesian and Galician material, and four parts chamotte. The charge of ore per 32 muffles is 2300 kg., of which the zinc tenor averages 20%. The recovery of metal is about 75%, the retort residues assaying 3 to 5% Zn. The crude spelter contains about 3% Pb, which is reduced to 1% by a subsequent refining. All the furnaces at the Hohenlohehütte are equipped with Kleeman condensers.

At the end of 1900 the Hohenlohehütte had 18 Freiberg roasting furnaces and 72 gas fired distillation furnaces of 32 muffles each,[1] a total of 2304 muffles. In 1900 the roasting plant desulphurized 39,335 metric tons of blende, yielding 33,419 tons of product. The consumption of coal for roasting was 15,101 tons, all of which was slack (Staubkohlen), or 38·4% of the weight of the raw ore. The labor cost was 166,296 marks for men (139), 152 for boys (1) and 15,146 for women (37), making a total of 181,594 marks, or 4·62 marks ($1·10) per ton of raw ore and 5·43 marks ($1·29) per ton of roasted ore. In the Hohenlohe smeltery there were treated 24,862 tons of roasted blende, 33,350 tons of calamine, and 1101 tons of between-products, a total of 59,313 tons, which yielded 10,296 tons of spelter and 452 tons of zinc dust; reckoning the latter as containing 90% Zn, or 407 tons, the total yield of zinc was 10,703 tons, or 18% of the weight of the ore smelted. The consumption of cinder was 32,283 tons, or 54·4% of the weight of the ore smelted. Of heating coal there were used 105,445 tons, chiefly small coal, which was 1·778 tons per ton of ore smelted and approximately 10 tons per ton of spelter produced. The consumption of fire clay was 6089 tons, of which 1562 tons were for making condensers, the total being approximately 0·1 ton per ton of ore. The number of muffles used was 33,572. Estimating the average working of the furnaces to have been 345 days this was equivalent to 97·3 per day, or 4·2% of the whole number in use (2304). On the same basis (345 days work) the average daily charge per muffle was 74·4 kg. and per furnace 2380 kg. The cost of labor was 723,994 marks for men (693), 8886 for boys (38) and 61,606 for women (178), a total of 794,486 marks, which was 13·4 marks ($3·19) per ton of ore and 77·1 marks ($18·35) per ton

[1] This refers to single furnaces, so called. The 64-muffle furnace is classed as a "double furnace." There would be therefore 36 double furnaces.

of spelter (10,296 tons). Besides the zinc dust previously referred to, there were produced 513 tons of lead and 4352 kg. of cadmium.

LETMATHE, WESTPHALIA.—At Letmathe, near Iserlohn, direct-fired furnaces with combined step grate and flat grate are used (1895). They have 75 elliptical retorts of 0·165 m. width and 0·255 m. height, inside measurements. The average charge per retort is 26 kg. of ore, chiefly roasted blende. From 38 to 40% of anthracite is mixed with the ore as reduction material. A 75-retort furnace distils 1950 kg. of ore per 24 hours, with a consumption of three to four tons of heating coal and 1·25 ton of reduction coal per ton of spelter produced. The breakage of muffles is 40 per 100 kg. of zinc. The loss of zinc in distillation is 14 to 16%. The furnace crew consists of one man working a 24-hour shift and two men working 10-hour shifts.[1]

MÜNSTERBUSCHHÜTTE, RHENISH PRUSSIA.—At the Münsterbuschhütte, at Stolberg, furnaces with heat recuperators for the secondary air only are employed. These furnaces have a gas producer at each end and 240 retorts, or 120 per side, arranged in three rows of 40 each. The retorts, which are 1·2 to 1·4 m. in length, are charged with 35 kg. of roasted ore and 14 kg. of coal; they are made of Belgian clay (60% chamotte and 40% raw clay), to which a small percentage of coke dust is added. The ore distilled assays 52 to 54% Zn. Lean coal is used as reduction material to the amount of 40% of the weight of the roasted ore. A charge is distilled in 24 hours. A 240-retort furnace reduces eight metric tons of roasted ore per 24 hours, with a consumption of 8·5 to 9·2 metric tons of coal in the gas producers. About 2·5 tons of heating coal and one ton of reduction coal are used per ton of spelter produced. Retorts last from 40 to 50 days, according to the percentage of iron in the ore distilled and the proportion of reduction coal employed. The furnace crew for 240 retorts numbers 14 men working eight hours, two stokers working 12-hour shifts, besides one laborer for wheeling ore, coal and residues.[2]

MULDNERHÜTTE, SAXONY.—The zinc smeltery at the Muldnerhütte at Freiberg, in Saxony, has a Siemens furnace with 32 muffles, which are 0·235 m. wide, 0·49 m. high and 1·58 m. long. Each muffle is charged with 50 kg. of roasted blende assaying 33% Zn and 25 kg. of coke, made from brown coal, as reduction material. One charge is distilled per 24 hours. In reducing 50 kg. of blende 1·5 hectoliter of heating coal is used. The weekly product of the 32-retort furnace is 2500 kg. of spelter and 250 kg. of blue powder, the latter assaying 90% Zn. Of the zinc content of the

[1] Schnabel, Handbuch der Metallhüttenkünde, II, 174.
[2] Schnabel, op. cit., II, 180.

ore 70% is recovered. The retort residues assay 10% Zn; they are argentiferous and are delivered to the lead smeltery for reduction in blast furnaces.[1]

The ore smelted at the Muldnerhütte is chiefly concentrated blende from the Freiberg dressing floors, which assays approximately 40% Zn, 20% Fe, 0·04% Ag and 30% S. It is roasted in Rhenania furnaces, which put through 8200 to 8400 kg. of ore per 24 hours with a consumption of 20 hectoliters of brown coal and 20 hectoliters of Bohemian soft coal, the former averaging 5 to 6% ash and the latter 15%. The ore is roasted down to less than 1% S. The distillation department used to have a Boetius furnace with 48 muffles (24 per side) and a Siemens furnace of 32 muffles (16 per side), but the Boetius furnace has now been abandoned in favor of the Siemens type.

NEUMÜHL-HAMBORN, WESTPHALIA.—At Neumühl-Hamborn, near Oberhausen, gas-fired furnaces with recuperation of heat in the air for secondary combustion are employed (1895). These furnaces have 252 retorts arranged in three rows of 42 each per side, i.e., 126 retorts per side. There is a gas producer at each end of the furnace. The retorts are 0·16 m. wide and 0·30 m. high and 1·4 m. long, inside measurements. They are made of a mixture of 60 parts burnt Belgian clay, 35 parts raw Belgian clay and five parts coke dust. The charge for distillation consists for the most part of roasted blende assaying 53 to 55% Zn, with which 40 to 44% of reduction coal is mixed, the latter consisting of coarse screenings of lean coal from the basin of the Ruhr. The charge per retort consists of 30 to 35 kg. of ore and 13 kg. of reduction coal, together with a small quantity of skimmings and condenser scrapings from the previous day. The total charge for a furnace of 252 retorts is 7600 to 8000 kg. of ore, exclusive of skimmings and scrapings. The distillation lasts 19 hours, five hours being occupied by the maneuver. The consumption of heating coal amounts to 12 metric tons per furnace per 24 hours; for this purpose a gassy coal slack with 20% ash is used; with good run of mine, such as is employed at other smelteries in Rhenish Prussia and Westphalia, the consumption is one third less. Per ton of spelter produced there are used 3·6 tons of heating coal and 0·88 ton of reduction coal. Retorts last 30 days on the average, eight being required per furnace per day. During the maneuver the furnace crew consists of 12 men; during the distillation there are two men, who work a 12-hour shift.[2]

PAULSHÜTTE, UPPER SILESIA.—This works, near Rosdzin, has 16 direct-

[1] Schnabel Handbuch der Metallhüttenkunde, II, 178.
[2] Schnabel, op. cit., II, 177.

fired furnaces of 32 muffles each; and 12 Siemens furnaces with 792 muffles in the aggregate. In 1895, according to Schnabel,[1] Siemens furnaces with only 32 muffles were in use, each furnace having one gas producer and one or two gas shafts. The gas producers were run with a fan blast of air, and the air for secondary combustion was also blown into the furnace. Muffles were made of 65% clay and 35% old material. The charge for distillation consisted of 30% roasted blende and 70% calcined calamine, cinder to the amount of 40% of the weight of the ore being used as reduction material. The charge per muffle amounted to 100 kg. On the average a 32-muffle furnace would put through 3000 kg. of ore, yielding 400 to 500 kg. of spelter, with a consumption of four metric tons of heating coal, i.e., 1·33 ton of coal per ton of ore, or 8 to 10 tons of coal per ton of spelter. The ore yielded 13% of its weight in spelter, the loss in distillation amounting to 20 or 21% of the zinc contents of the ore. The life of the muffles ranged from 40 to 50 days.

SILESIAHÜTTE, UPPER SILESIA.—At the Silesiahütte, at Lipine, in 1895, distillation furnaces with two gas shafts and 32 to 40 muffles (16 to 20 per side) were in use. Each furnace had one gas producer. The air for secondary combustion was blown in by means of a fan, mixing with the gas about 0·5 m. below the upper edge of the gas shaft. The muffles were made of 37 to 40 parts of raw clay and 60 to 63 parts of burnt clay shale and old material. The raw clay was obtained from Briesen, in Moravia, and Saarau, in Silesia; the clay shale from Neurode and Mittelsteine, in Silesia. The old material was pulverized to 6 or 7 mm. size. The charge for distillation consisted of 25 to 45 parts of calamine and 75 to 55 parts of roasted blende, varying according to the supply of the respective ores. Cinder to the amount of 45 to 47% of the weight of the ore was used as reduction material. The charge of ore per muffle was 90 kg. The period of distillation ranged from 16 to 17 hours, seven to eight hours being occupied by the maneuver. The life of the muffles varied from 35 to 47 days, depending chiefly upon the age of the furnace and therefore upon the greater or less solidity of its hearth and upon the percentage of roasted blende in the charge, charges high in blende requiring a higher temperature for distillation than calamine. The daily charge of a 32-muffle furnace was 2880 kg. of ore; of a 40-muffle furnace 3600 kg. The consumption of heating coal was 1 to 1·2 ton per ton of ore, or about 5·5 tons per ton of spelter produced. The heating coal was partly slack and partly a slaty lump coal. The consumption of cinder for reduction material was 0·45 to 0·57 ton per ton of ore, or about two tons per ton of spelter produced. On

[1] Handbuch der Metallhüttenkunde, II, 176.

the average two muffles were broken per ton of spelter produced. The yield of zinc was 20% of the weight of the ore, equivalent to 77 to 80% of the zinc contents of the latter, the loss in distillation being 20 to 23%.[1]

The distillation departments of the company owning these works are known as Silesia II and Silesia III. At the end of 1900 the former comprised 60 furnaces with a total of 1978 muffles, while the latter had 60, with 1984 muffles. All the furnaces were gas fired. The quantity of ore smelted in the two works in 1900 amounted to 51,910 tons of calamine, 65,904 of roasted blende and 401 of other products, a total of 118,215, which yielded 25,572 tons of spelter, or 21·6%. The consumption of cinder was 61,914 tons (0·52 ton per ton of ore) and of fuel of all kinds, but chiefly slack, 168,600 tons (1·426 ton per ton of ore=approximately 6·6 tons per ton of spelter produced). The consumption of fire clay was 3539 tons and of muffles 38,425, the latter figure indicating a breakage of approximately 2·8%. The average charge of ore per muffle was about 86·5 kg. The total cost of labor was 1,224,100 marks, or 10·35 marks ($2·46) per ton of ore=47·87 marks ($11·39) per ton of spelter produced.

According to memoranda made in 1893 all the furnaces of the Silesiahütte were equipped with Kleeman condensers. The furnaces were grouped in pairs with a kiln between them for calcining calamine, which was heated by the escaping products of combustion. The muffles, which were 0·625 m. high and 0·14 m. wide inside and 1·7 m. long, weighing 250 kg., were made of 60% chamotte and 40% raw clay. The raw clay was ground by means of coffee mill pulverizers and the chamotte by means of roller mills. The batch was mixed with 12% water and kneaded in vertical pug mills.

The blende which is treated at the Silesiahütte is roasted in various types of furnaces, including grate kilns, two-hearth reverberatories (Freiberger-öfen), Hasenclever-Helbig furnaces and Rhenania furnaces, the last being of the latest construction and the most preferred. A double furnace of that type, operated by eight men per 24 hours, will roast 8000 kg. of ore, containing 20 to 30% S down to 1% S, with a consumption of 20% of its weight of coal. The ore is crushed to pass a 2 mm. screen before roasting. The roast gases from the reverberatory furnaces are dispersed through a chimney 80 m. high; the sulphurous gases from the Rhenania furnaces, which contain 5% SO_2 by volume, are utilized for the manufacture of sulphuric acid.

WILHELMINEHÜTTE, UPPER SILESIA.—The ore treated at these works is a mixture of blende and calamine. The lump blende is roasted in kilns; the fines in two-hearth reverberatory furnaces. The roasted blende and cal-

[1] Schnabel Handbuch der Metallhüttenkunde, II, 176.

cined calamine are distilled in Siemens regenerative furnaces, of which in 1893 there were 30, having 68 and 72 muffles each, i.e., 34 and 36 per side. The average length of furnace campaign was said to be from three to four years. The muffles are 0·14 m. wide and 0·6 m. high inside and 1·7 m. long. They are made of 55% raw clay and 45% old material. A 68-muffle furnace is charged with 7000 kg. of ore assaying 20% Zn, with which is mixed 45% of reduction material. The yield of metal is about 1100 kg. which is a trifle more than 150 kg. per ton of ore, or 75% of its zinc contents. The distillation furnaces are fitted with Dagner condensers.

In 1900 there were in operation at the Wilhelminehütte 30 furnaces with a total of 2028 muffles, which smelted 67,139 metric tons of ore (calamine and roasted blende), yielding 11,743 tons of spelter, or approximately 17·5% of the weight of the ore. The total consumption of fuel and reduction material (cinder) was 105,124 tons, of which 4301 tons were for calcining calamine, 3300 tons for tempering muffles, 720 tons for refining crude spelter, 970 tons for cadmium smelting (4850 kg. of cadmium produced), 3710 tons for steam and power, and 87,827 tons for distillation. Estimating that the cinder consumed was 50% of the weight of the ore, the fuel alone would have been about 54,250 tons=approximately 0·8 ton per ton of ore and 4·6 tons per ton of spelter produced. In refining the crude spelter there was obtained 162 tons of lead, or 1·36%. The consumption of muffles for cadmium smelting was 135; for zinc smelting, 14,047. The latter figure indicates a breakage of 2% daily. The average charge of ore per muffle was approximately 96 kg. Labor cost nine marks ($2·15) per ton of ore=51·36 marks ($12·22) per ton of spelter produced.

Great Britain.

Swansea, Wales.—The English Crown Spelter Co. in 1892 smelted 9249 long tons of calcined calamine, 3147 tons of calcined blende, 681 tons of zinc ashes and 39 tons of flue dust, a total of 13,116 tons, which yielded 5527 tons of spelter. There were consumed 22,241 tons of coal for heating the furnaces and 6413 tons for reduction material, a total of 28,654 tons, besides which the requirements for other purposes raised the total consumption of the works to 32,124 tons. The consumption of retorts amounted to 12,293, and of condensers to 99,406, an average of 2·22 and 18 per ton of metal respectively. The total consumption of coal was 2·15 tons per ton of ore smelted and 5·81 per ton of metal produced. The coal for heating the furnaces amounted to 1·7 tons per ton of ore, while the coal used for reduction material was approximately 49% of the weight of the ore.

In 1895 the English Crown Spelter Co. smelted 13,350 tons of ore, of which 8753 were calamine, 4020 blende, 374 zinc ashes and 203 flue dust. The make of metal was 5727 long tons. The consumption of coal on the grates was 24,802 tons and for reduction material 6886 tons. The consumption of retorts amounted to 12,515. The consumption of reduction material was about 51% of the weight of the ore and of heating coal 1·85 tons per ton of ore. The ore distilled is of rather low grade. The distillation furnaces are partly direct fired and partly gas fired.

The average consumption of coal, retorts and condensers per long ton of metal produced in several years is shown in the following table:

	Coal *a*	Retorts	Condensers
1890	....	2·56	17·9
1891	5·03	2·26	17·7
1892	5·19	2·22	18·0
1894	....	2·31	16·9
1895	5·52	2·18	16·7

a Heating coal and reduction material; coal for other purposes amounts to about 0·6 ton per ton of metal additional.

EMU WORKS.—Considerable attention, especially in England, has been directed lately to the Picard and Sulman method of zinc smelting, which has recently been applied on a commercial scale at Cockle Creek, N. S. W., consignments of spelter therefrom having already been received in England. Before the erection of the works in Australia was undertaken the process was given a trial at the Emu works, in Wales, where upward of 4000 tons of ore were treated by it during 1901, so that it may fairly be considered to have passed the experimental stage. Its details were communicated by the inventors, Messrs. H. Kirkpatrick Picard and H. Livingstone Sulman, in a paper entitled "A Dry Process for the Treatment of Complex Sulphide Ores," read before the Institution of Mining and Metallurgy, June 19, 1902, of which paper the following paragraph is an abstract:

"The roasted ore is mixed with about 20% of its weight of crushed coking coal, and the mixture is briquetted in any suitable type of machine, pitch or some other carbonaceous material being employed as binding agent. The briquettes are then distilled in the ordinary manner and in the normal time. They coke into coherent masses and thereby form a skeleton, which holds up the particles of reduced lead and the corrosive matte and slag, and thus protect the walls of the retort. The distillation furnaces at the Emu works are of the direct fired, Welsh-Belgian type, with 144 retorts arranged in six rows, the lowest being cannon pots. The ore treated assays 25% Zn and

24% Pb; it is mixed with 20% of crushed coking coal and 5% of pitch. The residues assay 5 to 8% Zn. The recovery of zinc is about 70%. The consumption of retorts is 3·7 per furnace per day, the average life being 35 to 42 days. The retorts are made by hand and cost 6s. apiece, which is high even for Wales. The residues, coked briquettes, drawn from the retorts are smelted in the ordinary manner for recovery of their silver and lead contents. The loss of lead during the distillation of the zinc is insignificant and but very little lead goes over into the spelter, the latter averaging 99% Zn and only about 0·5% Pb."[1]

The paper of Messrs. Picard and Sulman failed to go into numerous details that we should like to know about. For example, the method of making the briquettes, their shape and physical characteristics, and the manner of charging them into the retorts are not described. It is only stated that a retort takes about 15 briquettes and the charging is effected by means of a shaped iron paddle. A furnace of 144 retorts takes seven tons of briquettes. The cost of briquetting 30 tons of roasted ore per day is 5s. 6d. per ton, or a total of 165s. 6d., of which six tons of coal at 8s. 6d. per ton account for 51s., and 1·5 tons of pitch at 45s. per ton come to 72s. 6d. Deducting the cost of the coal, which would be used as reduction material under any circumstances, the actual cost of the briquetting for labor and material is about 3s. 10d. per ton of roasted ore, but this is partially offset by the smaller percentage of coal employed.

The results reported by Messrs. Picard and Sulman are very interesting indeed because of the high tenor in lead of the ore distilled and the low percentage of reduction material employed. The latter is directly contrary to the approved practice in zinc smelting and would appear to indicate that the briquetting of the charge may be advantageous, both in preventing corrosive substances from reaching the walls of the retort and in increasing the capacity of the furnace.[2] It is upon this, indeed, that the only essential claim of the Picard and Sulman patent rests. United States patent No.

[1] The small percentage of lead in the spelter, together with the high percentage of zinc in the residues indicate distillation at a low temperature.

[2] These results may be accepted, but metallurgists differ as to the true reasons therefor. The metallurgists of the Overpelt works, Belgium, attribute them not to briquetting of the charge, but rather to the use of the pitch, the employment of which has been patented by Mr. Wilhelm Schulte (vide p. 505). F. Klessling, superintendent at Overpelt, in commenting upon the Picard and Sulman process (Berg-u. Hüttenm. Ztg., LXI, xxxviii, 483) states that very little lead remains in metallic form in the residues, the major part being slagged with silica, iron and lime. The briquettes come out in very fragile form, and to smelt them in a blast furnace without a previous sintering would be a doubtful process. The residues have a shining, black, graphitic appearance, which is probably due to the deposition in them of finely divided soot from dissociation of the hydrocarbon used as binding agent. In 1898 complex ore containing 25% Zn and 24% Pb was smelted by the ordinary method at the Birkenganghütte at Stolberg, Rhenish Prussia, without difficulty.

665,744, issued January 8, 1901, has a single claim, as follows: "The process of treating sulphide ores containing silver, zinc and lead, which consists in roasting the ore to the form of oxides, mixing the roasted product with carbonaceous material suitable for coking, forming the mixture into briquettes and distilling the briquettes under such condition that they are first coked into coherent masses and finally the zinc reduced and volatilized, while the lead is reduced and the lead and silver retained in minute particles throughout the coke." This does not, however, appear to be essentially different from the process of Binon and Grandfils, which is referred to in Chapter XI. The Binon and Grandfils process was tried practically at Bleyberg, Belgium, in 1882, with successful results it was claimed at that time, but the method did not continue in use.

United States.

Bertha Works, Pulaski, Va.—The ore smelted at these works is a concentrated calamine of approximately the following composition: 47·61% ZnO (38·08% Zn), 29·37% SiO_2, 9·23% $Fe_2O_3+Al_2O_3$, 4·54% $CaCO_3$, 2·07% $MgCO_3$, and 8·23% of combined water. It is dried in a revolving cylinder furnace, 30 ft. long, heated by an 8 ft. Taylor gas producer. The smeltery has 10 Welsh-Belgian furnaces, each containing 140 elliptical retorts, which are 4 ft. long and 8×10 in. in section inside. The heating coal is run of mine from the Pocahontas Flat Top region, 75 miles from Pulaski, which contains 74·27% of fixed carbon, 18·79% of volatile matter and 6·94% of ash. An anthracite coal from Altoona is employed as reduction material.[1] The ordinary charge per furnace consists of 4·25 tons of ore mixed with three tons of coal. The yield of spelter from such a charge is 1950 to 2000 lb., about 80% of the zinc in the ore being recovered. Prolongs are used on the condensers; the blue powder collected therein assays about 80% Zn. Most of the spelter produced at the Bertha works is exceptionally high grade, which has made its brand famous all over the world. In its production no iron tool is allowed to come into contact with the zinc in its molten condition.

Collinsville Works, Collinsville, Ill.—At these works the distillation furnaces are of the direct fired Belgian type, each furnace having 256 retorts arranged in eight rows of 18 each per side. The furnace cellars are on a level with the ground, the retorts being worked from an upper floor, which is reached by inclined planes. The ore smelted is exclusively blende from the Joplin district. It is roasted in Brown horseshoe furnaces, each

[1] The Altoona coal contains 62·72% fixed carbon, 10·52% volatile matter, 1·43% S and 25 to 33% ash.

of which puts through about 12 tons of raw ore assaying 28% S per day, yielding about 10 tons of product assaying 0·5 to 1% S. One man per shift (eight hours) attends to a furnace, doing the stoking and tramming the ore to it. The furnaces are fired with a very inferior slack coal, of which eight to 10 tons per 24 hours are required. The same kind of coal is used for heating the distillation furnaces. The latter have four grates, 7·5 ft.×15 in. There are only two bars per grate, a bed of clinker being carried on the grate to support the coal. The surface of the fire is maintained about 3 ft. above the grate bars. The furnace crew consists of one brigadier, two long shifts and two short shifts per 128 retorts. The brigadier and long shifts work 24 hours in the customary manner. The charge for 128 retorts consists of 5000 lb. of roasted blende mixed with 50% of reduction material (0·5 coke and 0·5 lean coal). The coke is made from coal that has been washed to remove the sulphur, but still contains 2·5% of that element. About 1800 lb. of blue powder and other between-products are made per day, which are put into the retorts of the seventh and eighth rows in charging the furnace a following day. The consumption of coal on the furnace grates is about 20 tons per 24 hours. The retorts are 8 in. in diameter inside and 10·5 in. outside; they are 48 in. long outside. They are annealed in down-draught kilns. About eight are required daily by each double furnace. The total consumption of coal is approximately eight tons per ton of spelter produced.

Columbia Works, Marion, Ind.—All the furnaces at these works are heated by natural gas. The ore is roasted in shelf burners with four hearths, each 7×8 ft.; the burners are built back to back in rows or massives. Each burner delivers two tons of roasted product per 24 hours, one man per shift attending to two burners. The work is done by contract at $1 per ton of product, at which rate the contractor earns about $2 per day.

The roasted ore is distilled in gas fired furnaces, of which the design has been referred to elsewhere in this treatise. The gas is introduced at 8 oz. pressure. The regular furnace charge is 4000 lb. of ore per 100 retorts, which yields about 2375 lb. of spelter. As reduction material there is employed a mixture consisting of 50% anthracite, 25% cinder and 25% lean coal. The recovery of zinc is about 82%, referring to the tenor of the roasted ore.

Edgar Works, Cherryvale, Kan.—The furnaces at these works are heated by natural gas. Brown horseshoe furnaces are used for roasting. The distillation furnaces have 200 retorts, arranged in five rows of 20 per side. The gas is introduced at 4 oz. pressure, through pipes which enter

the chambers from a central longitudinal tunnel under the middle wall of the furnace; gas is also admitted along the front of the furnace on each side through a horizontal pipe, of which the branches enter between the second and third row of retorts. With favorable ores the recovery of metal in distillation has been as high as 90%.

EMPIRE WORKS, JOPLIN, MO.—At these works, which are now idle, only blende concentrates from the Joplin district were treated. A specialty was made of buying some of the grades that contained more pyrites than was desired by other smelters, this material being roasted and then separated by means of Wetherill magnetic machines. The roasting at these works was done by means of a Wethey mechanical furnace, with hearth 164×12 ft., heated from fireplaces at the sides, with step grates and under-grate blast; and by a Pearce two-hearth turret furnace. The gases from these furnaces were discharged through a chimney 125 ft. high. The Wethey furnace was able to roast 20 tons of ore per 24 hours with two men per shift; the Pearce furnace would do about 10 tons of ore per 24 hours with one man per shift. The ore was roasted down to about 1% S. The labor and fuel cost was about the same per ton of ore with each furnace, but the repairs on the Wethey furnace were higher than on the Pearce. The distillation furnaces were of the direct fired Belgian type with 224 retorts. The regular charge was 4·8 tons per 24 hours and the average length of campaign 22 months. The retorts were cylindrical and of the regular dimensions; they were made by hydraulic pressure. All the machinery about the works was driven by electric motors fed by current from a central station. The ore was dried on inclined plates, heated by the gases from the roasting furnaces, whence it was moved automatically to the roasting furnaces. Each roasting furnace had its own drying furnace. The coal was also transported mechanically to the roasting furnaces.

GIRARD, KAN.—These works smelt zinc silicate ore assaying 42% Zn, of which five tons are distilled per 24 hours in direct-fired Belgian furnaces with 224 retorts. The yield of metal from such a charge varies between 1·4 and 1·5 tons, being equivalent to a recovery of about 70% of the zinc contents of the ore (1899).

GLENDALE WORKS, CARONDELET (SOUTH ST. LOUIS), MO.—At these works the ore treated is blende and calamine, but chiefly the former. The blende is crushed to pass punched steel plates with slots $\frac{5}{64}$ in. in diameter. It is roasted in hand-raked kilns with five hearths, which put through 13,200 lb. of raw ore per 24 hours, yielding 11,000 lb. of product, with two men per shift. A Brown horseshoe furnace is also used. It puts through from 11 to 12·5 tons of raw ore per 24 hours, roasting it down to 0·5% S in

the case of favorable ores. The furnace crew consists of two men per shift, one of whom manages the furnace and the other wheels coal, ore and ashes. The coal which is burned is slack of inferior quality; about 66⅔% of the weight of the raw ore is required. The draught for the furnace is furnished by a chimney 90 ft. high and 3×3 ft. in area at the top.

The distillation furnaces are direct-fired Belgians of various sizes and designs, according to their date of construction. Most of them have either 224 retorts (8×14 per side) or 256 (8×16 per side). The charge varies from 4·5 to 5·2 tons of ore, besides the blue powder and sweepings from a previous day. The ore is mixed with about 50% of reduction material, which is a mixture of coal and coke, or coal and cinder. The retorts are cylindrical, 8 in. in diameter inside and 10·5 in. outside. About 3 ft. of coal is carried on the grates, which are 6·5 ft.×16 in. The temperature maintained during the distillation period is 2300 to 2500° F. (1260 to 1270° C.). The coal and ore are conveyed to the furnaces by means of a single-rail overhead tramway, the buckets of which are pushed by hand.

Lehigh Works, South Bethlehem, Pa.—The production of spelter was begun at these works in 1858. In 1894 there were in use furnaces with 160 retorts, arranged in four rows of 20 each per side, and 280 retorts arranged in seven rows of 20 each. Previously all the furnaces had had seven rows, but it was found that the lower had to be butchered in order to heat the upper ones properly, wherefore the height of the furnaces was reduced. All these furnaces were divided into four sections by a middle longitudinal and middle transverse wall. Each furnace had three chimneys and four grates. The fuel burned on the latter was No. 2 buckwheat anthracite, a blast equivalent in pressure to a column of 4 in. of water being introduced under the grate. The average length of furnace campaign was said to be about 2·5 years. The retorts were 7 in. in diameter inside and 50 in. long. The ore distilled was entirely willemite, averaging about 50% Zn. When the tenor in iron and manganese was less than 11% retorts would last about 35 days on the average, but when iron and manganese exceeded 11% their life was reduced rapidly. Anthracite culm was employed for reduction material, about 40% of the weight of the ore being used. The recovery of zinc was about 82%, the retort residues assaying 6 to 7% Zn. The consumption of coal for heating the furnaces was about 2·25 tons per ton of ore.

Matthiessen & Hegeler Works, Lasalle, Ill.—The peculiar type of furnaces used at these works has been described in a previous chapter. The ore smelted is chiefly blende, which is obtained from Kansas, Missouri, Iowa and Wisconsin. The average grade is about 57% Zn before roasting

and 68% Zn after roasting, the percentages of iron being about 2 and 2·2 respectively. The ore is roasted in mechanically raked muffle furnaces, the sulphurous gas from which is employed for the manufacture of sulphuric acid. The ore is crushed to pass a 2 mm. round hole before roasting. The raw ore contains 29 to 30% S, which is reduced to about 1·2% in the roasted product. The roast gases contain about 5% SO_2 by volume. The roasted ore is mixed mechanically with 45 to 46% of anthracite culm for reduction material. The distillation furnaces are run without prolongs on the condensers, it having been proved by experiment that the saving effected with them was too small to compensate for the trouble. The retorts are made of fire clay from Cheltenham, Mo., about 44% raw clay and 56% chamotte, ground to 10 mesh size, being used. The clay is crushed in a roller mill. The batch is pugged in the ordinary manner, after which it is allowed to remain in a heap for a suitable time, and then is pugged again, after which the retorts are molded by means of an augur machine. The retorts are cylindrical, 8·25 in. in diameter inside and 54 in. long, the walls being 1·25 in. thick. The condensers are 24 in. long. An hydraulic press plant has recently been installed, making retorts of the same dimensions as before.

Passaic Works, Jersey City, N. J.—At these works the distillation furnaces have 216 and 252 retorts, the former number arranged in six rows of 18 each per side, and the latter in seven rows of 18 each. These furnaces are fired with small anthracite coal. Elliptical retorts are used in the lower four rows and cylindrical retorts in the upper rows. The average length of a furnace campaign is about two years. The ore smelted is New Jersey willemite together with calamine from Virginia. The charge averages about 45% Zn. Owing to its high tenor in iron and manganese retorts last only 26 or 27 days. The cylindrical retorts are 7 in. in diameter inside and 54 in. long, the walls being 1·25 in. thick and the butts 2 in. thick; the elliptical retorts are 7×9 in. inside and of the same length as the cylindrical. Both kinds are made of about 50% raw clay, and 50% chamotte and old material. Retorts are glazed both inside and outside. In the best practice at the Passaic works a saving of 83 to 85% of the zinc was effected, as a monthly average, but the general average was not so high (1893).

XVIII.

PROPOSALS TO SMELT ZINC ORE IN THE BLAST FURNACE.

Many attempts have been made to smelt zinc ore in the blast furnace, a process which is theoretically possible and is indicated practically by the occasional formation of metallic zinc as a by-product in smelting zinkiferous iron and lead ores. Any one who is interested in the records of the occasional and accidental formation of zinc in those processes can find abundant references thereto in metallurgical literature, especially in works pertaining to lead smelting in the Lower Harz, iron smelting in Upper Silesia, and the manufacture of spiegeleisen in New Jersey. Experiments for reducing zinc in blast furnaces were first undertaken by Adrian Müller at Gladbach in 1861. Furnaces for this purpose have been devised by A. Gillon (*Berg-u Hüttenm. Ztg.*, 1881, p. 6); L. Kleeman (German patent, No. 14,497; P. Keil (German patent, No. 15,992); G. Westmann (German patent, No. 19,127; F. Clerc (*Berg-u. Hüttenm, Ztg.*, 1877, p. 83; *Dingler's Polytech. Journ.*, CCXXIV, 179; *Iron*, 1876, p. 581); J. Glaser (German patent, No. 48,449; *Dingler's Polytech. Journ.*, CCLIV, 253); and C. Komarek (*Chem. Ztg.*, 1880, p. 135). Experiments have also been made by Dyar, Rochaz, Shear, Duclos, Schmelzer, Swindell, Broomann, Lencauchez, Lesoinne and many others. All these have been fruitless, however, and it is not worth while to relate them here in detail.

It has been found by all investigators of this subject that zinc vapor mixed with a large percentage of carbon monoxide and dioxide and nitrogen cannot be condensed to fluid zinc, the metal separating invariably as a powder. In attempting the reduction of zinc oxide in the blast furnace, the vapor is necessarily diluted largely with the combustion products of the fuel and the inert nitrogen of the air employed for its combustion. The gas is likely to be further diluted by the excess of air which can hardly be avoided. The carbon dioxide gas and the unconsumed air have an oxidizing effect not only upon the zinc vapor but also upon the zinc condensed in fine dust form, wherefore the product that is obtained is bound to be a mixture of fine metallic zinc and zinc oxide, unless the presence of these

oxidizing gases can be prevented. By the employment of hot air and an excess of fuel, however, and taking care that the ore shall contain neither water nor carbon dioxide, and selecting zinc ore free from oxides of iron (which are reduced in the upper part of the furnace by carbon monoxide, thus producing carbon dioxide) the oxidation of zinc can be considerably limited. It is possible therefore to produce from the blast furnace, either zinc oxide or blue powder containing only a small quantity of oxide, according to the conditions established in the furnace. This is indeed done practically on a large scale in smelting zinkiferous manganese residues for spiegeleisen in New Jersey. In the present state of our technical knowledge, nothing more can be expected, unless the proposals to smelt under pressure, or with the use of water gas as reducing agent, which have lately been made and will be referred to further on, lead to some practical result.

The production of zinc powder on a large scale in the blast furnace requires great care in the prevention of oxidizing influences, not only to produce it as such, but also to preserve it. The metal of blue powder is in such an excessively fine state of subdivision that it is extremely susceptible to oxidation, so much so as to be explosive. On that account great precautions must be taken in its production. Only a few years ago there was an explosion of blue powder in the condensers connected with the spiegeleisen furnaces of the New Jersey Zinc Co., at Newark, N. J., which resulted fatally to one or more laborers who were working at it. Great care must be taken even in the storage of blue powder, owing to its liability to spontaneous combustion because of the large quantity of air that is held in its interstices; it is rated by fire underwriters as a specially hazardous risk and city ordinances specify that it may be stored only by special permit.

The oxidation of zinc by carbon dioxide was studied by Lencauchez, a distinguished French engineer,[1] who found that zinc was still oxidized when the proportion of carbon dioxide was only 0·5% of the gas employed. By very rapid cooling of the gases Lencauchez succeeded in reducing the oxidation of the zinc to as little as 5%, but after numerous experiments he concluded that the project of smelting zinc ore in the blast furnace was not likely to be commercially successful.

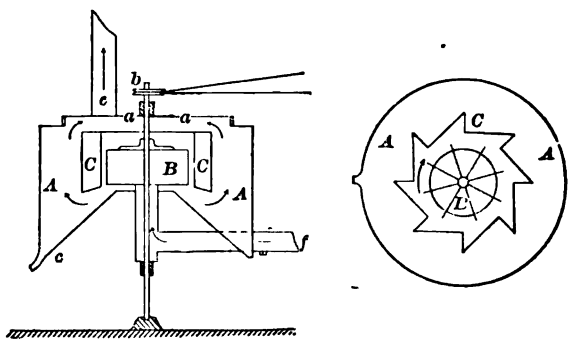
The reduction of zinc ore proper in the blast furnace is a theme which merges into the subject of the treatment of mixed sulphide ores, inasmuch as a method which would enable a successful distillation of a high grade zinc ore to be effected in that manner would probably be applicable in the same manner, or a modification of it, to the reduction of a mixed ore.

HEMPEL'S EXPERIMENTS.—The most recent and perhaps the most prom-

[1] Mém. Soc. Ingén. Civils, 1877, 568; Proc. Inst. Civil Engineers, LI, 332.

ising experiments on the smelting of zinc ore in the blast furnace were made by Professor Walther Hempel,[1] who showed that by the use of a hot blast it is possible to obtain a rich zinc fume which can be separated from the diluting gases of combustion by means of centrifugal force. The fume thus obtained is capable of compression into a very small volume, in which manner it can be protected from oxidation.

Hempel prepared the charge for his furnace by heating a mixture of one part zinc oxide, three parts of coal and 0·05 part of caustic lime in a retort, and allowing the coked mass to cool with exclusion of air. The blast furnace was designed like a Sefström furnace and was provided with an iron stove for heating the blast. The gas drawn off from the throat of the furnace was made to pass through an iron pipe, in which it was cooled down to 30° C. The exhaustion of the gas from the furnace and the sepa-



FIGS. 401 AND 402. HEMPEL'S APPARATUS.

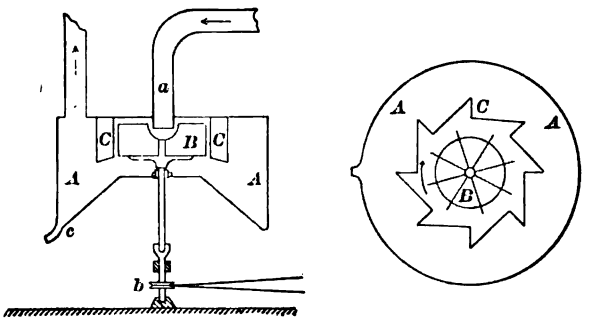
ration of the fume was effected by means of a centrifugal fan which is shown diagrammatically in the accompanying engravings. The fan wheel had eight blades and was driven at 1000 to 3000 r. p. m. It rotated inside of a casing *C*, which was contained inside of the cylinder *A*. The casing was open at the bottom but closed at the top, while the cylinder was closed at the bottom in the form of a frustrum of a cone and had a removable cover *a* at the top. The gas entered through the pipe *f* and was flung by centrifugal force against the projections of the casing *C*, whence the dust dropped down into the hopper of the cylinder *A* and settling there could be removed by the spout *c*. The gases escaping through the pipe *c* were passed through a bag-filtering apparatus in which any zinc dust escaping the centrifugal machine was collected. The percentage of carbon dioxide, which decreases as the furnace temperature increases and amounts to less than 1% at white heat, had gone down to 0·7% before charging the furnace and

[1] Berg-u. Hüttenm. Ztg., 1893, Nos. 41 and 42.

during the three hours that the distillation lasted, varied between 1·8 and 4% of the products of combustion.

The zinc fume which was produced assayed 72 to 90% Zn. It could be reduced by a pressure of 30 atmospheres to 13·3% of its original volume, to 10% by a pressure of 100 atmospheres and to 8·7% by a pressure of 200 atmospheres. The distillation of the condensed fume, without addition of carbon, yielded two thirds of its weight in the form of very pure spelter.

In treating a charge in the blast furnace to which ferric oxide and argentiferous lead had been added, Hempel obtained a fume assaying 80% Zn which contained the whole of the lead and the silver of the charge, while the iron was recovered in the form of pig and the gangue was fused as a slag assaying 58·3% SiO_2, 10·4% Al_2O_3, 8% Fe_2O_3, 15% CaO, 1% Zn and 1·8% S. The fume after compression into briquettes was distilled without addition of carbon, yielding a pure spelter to the amount of 66·67%



FIGS. 403 AND 404. HEMPEL'S APPARATUS.

of its original weight and leaving a residue in the retort assaying 41·6% SiO_2, 2·93% Fe_2O_3, 0·6% CaO, 33·6% Zn, 8·1% S and 1·05% PbS+ Ag_2S. As an alternative to the distillation, Hempel proposed to compress the fume into anodes for electrolysis, the zinc of which being largely in metallic form would develop a considerable quantity of electrical energy by its solution, so that only a low current tension would be required.

Proposed Process for Mixed Ores.—On the basis of these experiments Hempel proposed a method for the treatment of mixed ores, sulphide or oxidized. The ore after a preliminary roasting or calcination is to be sintered, or if coking coal be available, mixed therewith without previous sintering and coked. If the ore be high in iron, the oxides of iron must first be reduced, since otherwise zinc vapor would be reoxidized by the carbon dioxide produced in the upper part of the furnace. The manner in which this reduction is to be effected is not described in the original paper. After

these preliminary treatments the ore is to be smelted in a blast furnace, and the fume separated from the gases of combustion by means of a centrifugal apparatus and filtration system, the pure gas from the latter being valuable as a fuel gas or for the operation of gas engines, which experience in the utilization of iron blast furnace gas has shown to be perfectly feasible. The gas leaving the furnace is first passed through iron pipes exposing sufficient surface to cool it down to 50° C. A portion of the fume is deposited in the pipes. The cool gas is passed through a series of centrifugal machines, in which nearly the whole of the zinc fume will be separated. Instead of the machines illustrated in Figs. 401 and 402 it is proposed to design them as shown in the Figs. 403 and 404. These fans will be driven from below and the dust-laden gas will be introduced at the top. If the speed required for complete separation of the fume be found to be too high, the fans will discharge into a bag-filtering system to separate the last of the fume.

In order to avoid the formation of zinc oxide the blast furnace should be worked so that the tenor of CO_2 in the gas should not exceed 4%. The collected zinc fume will be compressed by a hydraulic apparatus into cylindrical blocks of a size corresponding to the retorts of the distillation furnace.

Hempel's process has not yet been attempted in practice, but his experiments appear to have been made in a careful manner and their results deserve consideration. No data were given as to the amount of power required to drive the fan, but fans of large size are obviously required to handle the great volume of gas that would be produced by a blast furnace, and in driving it at the very high speed proposed, the consumption of power would necessarily be large.

LUNGWITZ'S PROPOSAL.—Doctor Emil E. Lungwitz has proposed to smelt zinc ore in the blast furnace under a pressure higher than that at which the metal would boil at the temperature attained in the furnace.[1] It is well known that the boiling point of a liquid depends upon the pressure exerted upon it. Sufficient pressure will prevent ebullition of a liquid even if its temperature be raised considerably beyond its boiling point under ordinary pressure. In order to prevent volatilization, therefore, according to Doctor Lungwitz it is only necessary to maintain a sufficiently high pressure in the furnace. In the case of an ore containing lead and zinc oxides, the zinc would volatilize at 1200° C., if a sufficient pressure were not exerted to raise its boiling point to that temperature. He states that from the result of his own investigations and those of Doctor Carl Barus[2] it may

[1] United States patent, No. 538,785, May 7, 1895; 555,961, March 10, 1896.
[2] Bulletin of the United States Geological Survey, No. 54, 1889.

be concluded that the tension of zinc vapor at that temperature would be about 50 lb.; at 1000° C. it is less than 30 lb. (two atmospheres). Doctor Lungwitz, in the patents referred to above, described a furnace for carrying out his idea, which in itself is undoubtedly sound, although it is questionable if the particular furnace outlined for the purpose would be satisfactory in practice.

Acting upon the above idea, Doctor Robert C. Schüpphaus described in *Journ. Soc. Chem. Ind.*, Nov. 30, 1899, p. 987, a series of experiments as to the behavior of zinc oxide at high temperature under pressure. An electric furnace with a cast iron melting pot, lined with fire clay, of sufficient strength to withstand a pressure of 10 atmospheres, was used. It was found that the reduction of zinc oxide by carbon began at a temperature of 910° C. In heating charges of zinc oxide and carbon up to 1150° C. and maintaining a pressure in excess of the calculated tension of zinc vapor (two atmospheres), ingots of zinc were obtained at the bottom of the crucible, although its temperature was far beyond 920° C., the boiling point of zinc under atmospheric pressure. This method of smelting is suggested, among other things, for the production of spelter from blue powder.

Other Recent Proposals.—Numerous patents are taken out annually for processes of smelting zinc ore in the blast furnace; it is scarcely worth the trouble to maintain a complete list of them. Some of the more recent proposals are as follows:

Armstrong's Process.—J. Armstrong, of London, England, patented a blast furnace for smelting zinc ore wherein the volatilized metal is caused to pass through incandescent fuel and then through a bath of molten zinc maintained at a temperature a little above the melting point. The furnace proposed for this purpose resembles an ordinary circular lead-smelting blast furnace, of which the shaft is divided down to the bosh into a central cylindrical portion and an annular portion surrounding it, the tops of both being capable of being closed tightly. The charge of calcined ore, mixed with reduction material, is fed into the central portion, while carbonaceous fuel is introduced into the annular portion of the shaft. The two columns in descending unite in the bosh of the furnace, which is water-jacketed in the usual manner. The products of combustion, together with the volatilized zinc, are taken off through ports in the side of the annular shaft, just above the water jackets, and in order to escape must pass through a bath of molten zinc contained in a trough surrounding the furnace, which naturally traps the gas outlet of the latter. There are tap-holes in the trough for drawing off the condensed zinc when the latter exceeds the desired level. The non-volatile metals are reduced and together with the slag run down

into the crucible of the furnace below the level of the tuyeres, whence they are tapped in the ordinary manner.[1]

In a subsequent patent Armstrong proposes to collect the zinc as dust in an atmosphere of carbon monoxide proceeding from the furnace by passing the gas and fume through a column of incandescent coke and thence through cooled pipes to a closed condenser which is trapped against escape of the metallic dust by means of a seal of heavy oil, or other suitable liquid. The collected dust is compressed into blocks and after admixture of carbon is distilled, the vapor being received in a bath of liquid metal as described in British patent No. 3462 of 1900.[2]

Blenkinsop's Process.—G. H. Blenkinsop, of Swansea, proposes to smelt zinc-lead ores in blast furnaces, with or without flux and using a hot blast, taking care that the gases drawn off from the throat of the furnace are always at a temperature above that at which zinc vapor condenses. The volatilized zinc is to be collected in suitable condensing chambers.[3] There is nothing novel or useful in this idea.

Biewend's Process.—R. H. T. Biewend has proposed to smelt zinc blende in the blast furnace and effect the decomposition by means of iron, added either as metal or as oxide charged for that purpose. Sufficient fuel is to be used to reduce carbon dioxide to monoxide. The zinc vapor is to be conducted at a temperature of 800° C. into condensers filled with red-hot coke, wherein it is expected that it will be condensed in liquid form. The sulphide of iron produced by decomposition of the blende, together with the slag arising from the gangue of the ore and suitable fluxes, is to be tapped off from the bottom of the furnace. The iron sulphide can be reconverted into ferric oxide by calcination, the sulphurous fumes being available of course for the manufacture of sulphuric acid.[4]

In later patents Biewend undertakes to circumvent certain of the practical difficulties in smelting zinc ore in the blast furnace, which have been referred to in the early part of this chapter. His idea is to enrich the vapor issuing from the throat of the furnace by revolatilization of dust previously deposited. The process comprises the use of condensers filled with porous or rough bodies, or provided with other devices that serve to dry-filter the gases, the arrangement being such that the gases are caused by suitable reversals to pass alternately in opposite directions through the chambers. When a certain quantity of dust has been deposited a valve is operated to reverse the direction of the gases in their passage through the chamber.

[1] British patent, No. 3462, Feb. 21, 1900; Journ. Soc. Chem. Ind., April 30, 1901, p. 367.

[2] British patent, No. 11,339, June 3, 1901.

[3] British patent, No. 8990, April 8, 1897.

[4] German patent, No. 81,358, August 7, 1894; Zts. für angew. Chem., 1895, p. 358.

The gases then pass at their highest temperature and with the highest percentage of contained zinc into that part of the chamber which has, up to now, been the hindmost part, and impinge upon the dust that was therein deposited, and act to revolatilize it and by agglomeration with fresh globules of zinc to convert the dust into drops. Again, after a certain period has elapsed, there will be deposited at the other end of the chamber a sufficient quantity of zinc dust which is recovered in the same manner—that is to say, by again reversing the gases, namely, by causing them to pass again in their original direction. By this reversal of the direction of flow of the current of gas being repeated at certain intervals of time, too great a deposit of dust which would otherwise choke up the chamber is prevented, and the gases are also enriched by means of the dust which is deposited at their point of entry into the condenser.[1]

Biewend's process has not yet found practical application.

Nagel's Process.—Oskar Nagel, of New York, reports that he has found in his experiments that zinc vapor diluted with nitrogen will always condense to dust instead of to liquid metal, and, moreover, that if in addition to the inert gas there be present oxidizing agents, such as carbon dioxide, a corresponding part of the dust will be immediately oxidized. If inert and oxidizing gases be excluded from the furnace the zinc vapor may be condensed to spelter. To meet this condition he has devised the process which consists in distilling ore mixed with coal in an atmosphere of water gas, this being an active reducing agent, preferably heated before being introduced into the furnace, thereby doing away with the necessity of using other fuel. The coal mixed with the ore is employed only for the purpose of reducing any gases which may become oxidized by the reduction of the zinc oxide. It is claimed that zinc vapor can thus be obtained in as concentrated condition as in the ordinary process of distillation, and condensation to spelter can be readily effected, a yield of 90 to 95% of zinc being obtainable.

The water gas, which contains approximately equal volumes of carbon monoxide and hydrogen, having been prepared by well known methods, it is heated in ovens by means of the waste reduction gases coming from the condensation apparatus. The water gas is preferably forced into the furnace by means of a blower. The furnace in which the reduction is effected may be constructed similarly to the ordinary blast furnace for lead smelting, or upon the lines of furnaces used for distilling mercury. It should

[1] British patent, No. 30,152, Dec. 31, 1896. "Procédé de fabrication du zinc au moyen des gaz zincifères, et son application aux procédés connus," by R. H. T. Biewend, Moniteur Scientifique, February, 1898, p. 180.

be provided with the usual charging hopper and with a slag tap at the bottom. The tuyeres should be located about one third of the height of the furnace from the bottom. A firebrick lined flue leads out of the furnace near its top, through which the vapor is conveyed to the condensation apparatus. With a furnace of capacity for 25 tons of zinc oxide per 24 hours, about five tons will be reduced in four and a half to five hours, the water gas being introduced at temperature of about 1000° C. It is said that in practice it is found that 1·5 tons of water gas containing 1·4 tons of carbon monoxide and 0·1 ton of hydrogen, occupying a bulk of 10,415 cu. m. at 1000° C., will be sufficient for the reduction of 8·1 tons of zinc oxide.[1]

PRODUCTION OF ZINC OXIDE IN BLAST FURNACES.—The production of zinc oxide in blast furnaces is accomplished more easily than the production of metallic zinc, but for this purpose a grate kiln like that of Wetherill, which is described in Chapter XIX, has been found practically to be preferable to a shaft furnace. The residue of the ore burned in such kilns is smelted in a blast furnace by a subsequent operation. Ellershausen has succeeded, however, in effecting the volatilization of the zinc and the smelting of the residue in one operation in a blast furnace.

In producing zinc oxide in a shaft furnace, its formation is promoted by blowing an excess of air into the furnace, so as not only to insure the presence of an ample supply of free oxygen with which to burn the reduced zinc, but also to effect complete combustion of the coal so that the powerful oxidizing agent, carbon dioxide, will be produced instead of carbon monoxide. The formation of carbon dioxide is promoted, moreover, when the ore contains ferric oxide, which is reduced by carbon monoxide with the formation of carbon dioxide. Under such conditions the zinc reduced by carbon and carbon monoxide in the lower part of the furnace is oxidized by carbon dioxide and excess of air in the upper part. Such a smelting is naturally effected best in a low furnace with a light column of charge. That it can be done successfully has been demonstrated by the experience of F. L. Bartlett at Cañon City, Colo., and by F. Ellershausen at Angoulême in France.

[1] United States patent, No. 699,969, May 13, 1902.

XIX.

MANUFACTURE OF ZINC DUST, ZINC WHITE, ZINC SULPHATE AND ZINC CHLORIDE.

The chief forms in which zinc is prepared for market, otherwise than as spelter, are zinc gray, zinc white, zinc sulphate, zinc sulphide (lithophone) and zinc chloride. Zinc white and zinc sulphate are made either from spelter or directly from zinky by-products and ores; the methods employed in their manufacture from ores are of great interest to the zinc metallurgist. Zinc chloride is commonly made from spelter or sheet zinc scrap. Lithophone is made indirectly from ores through the intermediary product of zinc sulphate. Zinc dust is a between-product formed unavoidably in the smelting for spelter.

Zinc Dust.

The product which is known in commerce under the name of zinc dust is the blue powder of the smelters; it is also known as indigo auxiliary. Although every zinc smeltery makes it as an unwelcome but unavoidable between-product, it is only in Europe that it is prepared especially for sale, and the American requirements of the article are supplied from abroad. The method of preparation is extremely simple, consisting merely in collecting the blue powder from the prolongs, as free from admixture with foreign material as is possible, and sifting it. Since its value is dependent upon its tenor in metallic zinc, and is deteriorated by the presence of zinc oxide, it is essential in preparing it for market to manage the distillation and condensation of the zinc so that there will be the least possible formation of oxide. The market standard is a tenor of at least 90% metallic zinc.

Zinc dust is produced by numerous works in Upper Silesia, Westphalia, Rheinland and Belgium. There are no complete statistics as to the total production. The make of the works in Upper Silesia was 1093 metric tons in 1896; 993 in 1897; 886 in 1898; 915 in 1899, and 1472 in 1900.

Its selling price at Breslau is about 90 to 95% of the price of spelter, but in the United States it commands a premium over the price of ordinary spelter, although by a decision of the United States General Appraisers at New York, August 6, 1900, the article is admitted into the United States as a crude material, duty free. Zinc dust is marketed in the United States in kegs containing 500 lb. and barrels containing 1500 lb. Its value at New York in April, 1901, was 6·75c. per lb. The price of zinc dust fluctuates more or less according to the price of spelter, but is affected also by the demand and the ability to obtain storage for it, which in New York and other cities is governed by municipal ordinances and the regulations of fire insurance companies. The danger of explosion of zinc dust was referred to in Chapter XVIII.

A system for collecting in the form of dust the entire quantity of zinc distilled from a charge was recently patented by George G. Convers and Arthur B. De Saulles, of South Bethlehem, Penn.[1] This consists in the use of a distillation furnace having a single combustion chamber, in which the retorts set horizontally. The retorts are open at each end. One end, being that through which the charge is introduced, is luted up tightly during the distillation; the other end communicates, by means of a nipple, with a vertical, sheet iron canister, which acts as the condenser. In the front of the condenser there is a hole, closed ordinarily by means of a slide, which corresponds in position with the hole communicating with the retort. During the period of the distillation when the blue powder is likely to be contaminated largely with oxide, namely, at the beginning and the end, a small sheet iron cone is passed through the opening in the front of the condenser and connected with the nipple of the retort. This catches the zinc oxide separately. As soon as the color of the deposit at the outer end of the cone indicates the deposition of blue powder, the cone is removed and the slide in the front of the condenser is closed. The drawings accompanying the patent specifications show this type of condenser and form of furnace construction adapted to a furnace with one row of retorts, and to another with several rows. The size of the condenser depends naturally upon the richness of the charge and the volume of vapor to be passed into it. According to the inventors, a condenser 4 ft. in height and 2 ft. in diameter will collect 260 lb. of zinc dust per 24 hours. The temperature maintained in the condenser is from 320° to 415° C.

Zinc dust is employed chiefly as a reducing agent in indigo dyeing, and to a less extent finds use in the manufacture of paint, especially for covering iron work, and as a substitute for zinc shavings in the precipitation of gold

[1] United States patent, No. 695,376, March 11, 1902.

from the solutions obtained in the cyanide process, about 420 g. of zinc dust being required for each ounce of gold to be precipitated.[1]

Zinc White (Zinc Oxide).

In Europe this valuable substance, which is by far the most important of the compounds of zinc, is made exclusively by the oxidation of spelter. In the United States a little is made in that manner, but the more part of the large production is derived directly from ore. Recently a large export trade in the product thus prepared has been established. One of the most important recent developments in the American zinc industry has been the remarkable increase in the production of zinc oxide. It is said that this has been partly absorbed by an increased consumption in the rubber trade, zinc oxide being an important ingredient in the compounding of rubber; it has partly been exported, there being a growing foreign trade in the American product; but notwithstanding those demands it must have gained great headway in the domestic paint trade in competition with the other white pigments, especially white lead, which appears to be gradually falling into disfavor as a pigment when used unmixed with other substances.

The merits of zinc white as a pigment were summarized by Maximilian Toch in a paper recently read before the New York Section of the Society of Chemical Industry.[2] He stated that the consumption of white lead is decreasing and that it should not be used alone. The specifications of the United States Lighthouse Department demand a mixture of 75% of zinc oxide and 25% of white lead. Zinc white mixes well with other pigments, particularly with white lead. Zinc oxide made directly from ore is more durable than that made by burning spelter, and on a very large surface the difference in whiteness is soon apparent. Zinc oxide as the drying progresses becomes very hard and brittle, this being more the case with oxide made from spelter than with that made directly from ore. This is overcome by treating the linseed oil with 0·33% of litharge before mixing with the zinc white, a flexible drying paint being thus produced, which is essential in the manufacture of oilcloth. Zinc oxide is not affected by hydrogen sulphide, which gives it a superior advantage for the painting of the interiors of buildings. It is less absorbent than other paints, and can be washed. On outside walls it well withstands the action of the atmosphere when used to the extent of 50% in admixture with white lead and barium sulphate.

Manufacture Direct from Ore.—Zinc oxide is manufactured in the United States almost exclusively by the Wetherill process. There are four

[1] The Mineral Industry, VIII, 647.
[2] Journ. Soc. Chem. Ind., Jan. 31, 1902, p. 102.

works engaged in the business in New Jersey and Pennsylvania, all of which are operated by the New Jersey Zinc Co., and one at Mineral Point, Wis., operated by the Mineral Point Zinc Co., which is affiliated with the New Jersey Zinc Co. Page & Krause, of St. Louis, make about 500 tons of zinc oxide per annum, and the Standard Oil Co. recovers a little as a by-product at Williamsburg, Brooklyn, N. Y. The only other producers in the United States are the American Zinc-Lead Co., of Cañon City, Colo., which makes a zinc-lead pigment, and the Renfrow Paint Co. and Ozark Zinc Oxide Co., of Joplin, Mo., which employ the calamine of that district.

The Wetherill process is one of the most important contributions that America has made to the metallurgy of zinc. It was introduced as early as 1856, but remains essentially the same to-day as it was forty years ago, and since twenty years at least there has been no important metallurgical change in it. There has been, however, in New Jersey and Pennsylvania, a radical industrial innovation in the character of the ore sent to the oxide furnaces, which was brought about by the Wetherill process for the separation of franklinite and willemite. In making zinc oxide by this method from blende the latter is first roasted. When the oxidization of the sulphides and the volatilization is performed in the same furnace, as in the Bartlett process, the fume contains sulphates and complex sulphur compounds. Zinc oxide used to be made from roasted blende at Bergenpoint, N. J. (the ore being obtained from Pennsylvania) and is now so made at Mineral Point, Wis. In Europe this process has not found any important application.

The Wetherill process of making zinc oxide, as practiced in New Jersey and Pennsylvania, consists essentially in the reduction and distillation of the zinc contents of an oxidized ore, mixed with a certain quantity of anthracite as reducing agent and spread over a layer of fine anthracite, which is burned with the aid of an underblast on a perforated grate and furnishes the heat for the reduction. The volatilized zinc vapor is immediately oxidized, and together with the gases of combustion is drawn from the furnaces, cooled in a properly designed system of flues and chambers, and finally passed into a series of muslin bags, through which the gases of combustion escape while the condensed zinc oxide collects inside, whence it is removed at proper intervals. This system of fume filtration is the prototype after which the Lewis & Bartlett method of manufacturing lead pigment, the F. L. Bartlett process of making zinc-lead pigment, and the system of condensing lead fume escaping from the blast furnaces, which is employed at the Globe plant of the American Smelting & Refining Co., and at the Alton works of the Federal Lead Co., have all been designed.

Wetherill Furnaces.—The Wetherill furnaces are simple kilns of brick,

with arched roofs and suitable working doors in each end, and a grate area of 4×10 ft. per kiln. The grate is composed of flat cast iron bars 5 ft. long, 6 in. wide and 1 to 1·5 in. thick, perforated by tapering holes of 1 in. diameter on the lower side of the bar and 0·25 to 0·40 in. on the upper, which holes are made as closely together as can be done without taking away too much from the strength of the bar. Generally there are about 100 holes per square foot. The bars rest on bearers set crosswise into the side walls of the furnace, a single kiln requiring 16 bars. The arch of the roof, which has a 4 ft. span, begins 10 in. above the grate and rises to a height of about 3 ft. above the latter. Below the grate there is an ash pit 20 in. deep which is closed by iron doors. The ash pit is connected with an air flue,

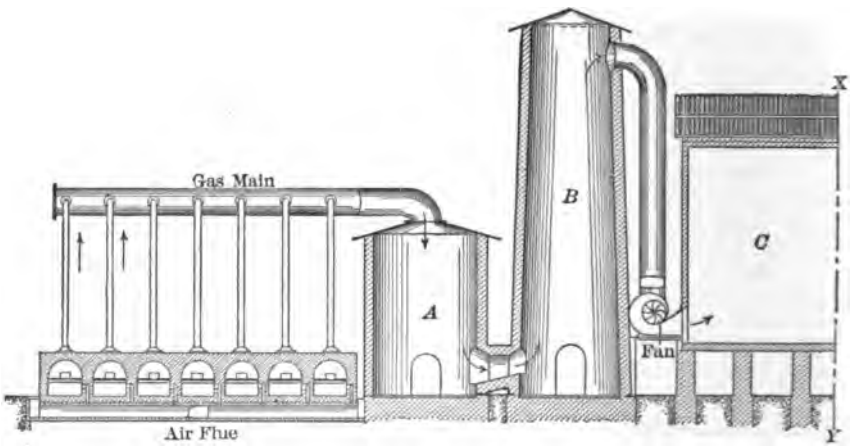
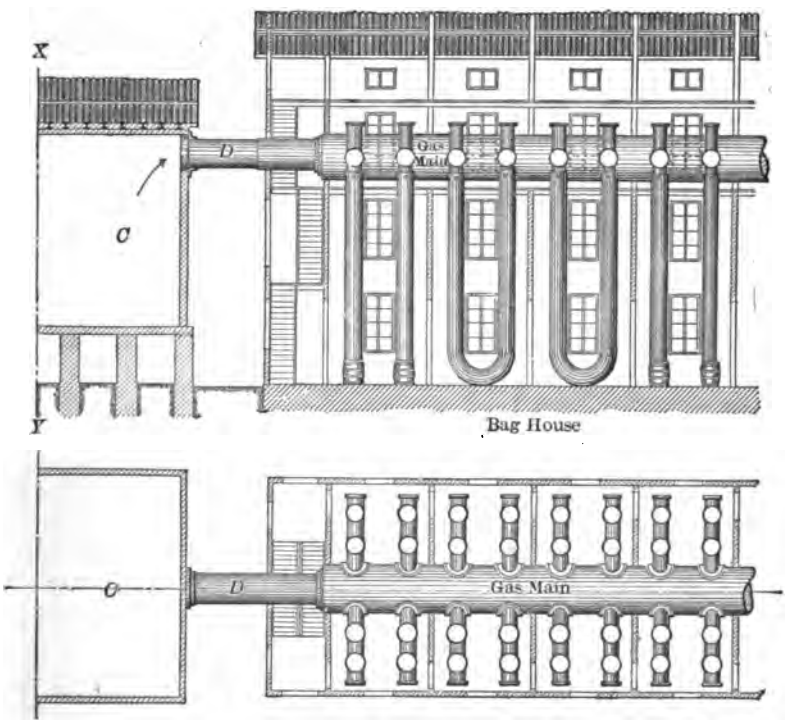


FIG. 405. ARRANGEMENT OF PLANT FOR MAKING ZINC WHITE.
Longitudinal elevation.

extending longitudinally under the block of furnaces, from which branches lead upward in the walls dividing the various ash pits; each branch flue is provided with an adjustable gate for regulation of the air supply, which is furnished by centrifugal fans. The air (cold) is blown in at a pressure of from 2 to 4 in. of water (1·16 to 2·31 oz.). There is an opening in the roof arch of the furnace for the escape of the fume and the products of combustion. The older furnaces had a single opening, 12×12 in., communicating with a longitudinal main flue, built on the top of the block; the later furnaces have two openings, through which the gas is conducted by means of pipes to a main as shown in Figs. 405 and 406. Both the front and back walls of the furnace have an opening above the grate for charging

and discharging, which opening is closed either by an iron door, or by piling in it a bank of wet ashes; for the latter purpose a shelf about 20 in. wide projects out from the furnace on a level with the grate bars. The front opening is commonly a semi-circular arch the full width of the furnace, or nearly so. A series of six furnaces is usually built in a row, the parting wall between adjacent furnaces being common to both of them; it was formerly customary to build 12 furnaces in a block, i.e., two rows of six



FIGS. 406 AND 407. ARRANGEMENT OF PLANT FOR MAKING ZINC WHITE.

Fig. 406: Longitudinal elevation. Fig. 407: Plan. Fig. 406 is a continuation of Fig. 405, joining on the line *XY*.

each, the two rows being set back to back, under which arrangement each furnace was 3·5×6 ft. in size and was of course charged only from one end. In later constructions the middle parting wall was left out, making six furnaces, 3·5×12 ft., which dimensions were subsequently modified to 4×10 ft. At Palmerton, however, the furnaces have been installed back to back in blocks of twelve.

The Palmerton plant has 24 blocks of 12 furnaces each, a total of 288

furnaces, which are installed in a building 50 ft. wide and 1260 ft. long. The furnace house is constructed entirely of iron and steel. The working floor is 10 ft. above the ground level, enabling the residuum drawn from the furnaces to be dropped into hoppers below the floor, whence it is received in steel cars, which are trammed by locomotives to the spiegeleisen blast furnaces. Each block of oxide furnaces is an independent unit. The blowers are situated in separate houses, of which there is one for every two blocks, each house containing two blowers, which are driven electrically by directly connected motors. The elevation of the working floor is an improvement over the arrangement of the older works, wherein the furnaces were set directly on the ground level.

Method of Operation.—In the operation of the furnaces the heap of ashes closing the front door is removed after a charge of ore has been worked off and is thrown to one side for use again. The residuum from the charge, which is in the form of a more or less solid clinker, covering the whole sur-

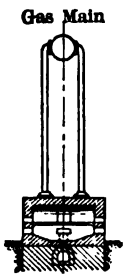


FIG. 408. ARRANGEMENT OF PLANT FOR MAKING ZINC WHITE.

Transverse section through furnace.

face of the grate, is then barred up and raked out through the doors. This manganiferous residuum goes to the blast furnaces for smelting to spiegeleisen. The grate having been cleaned, a layer of anthracite, nut size, is spread evenly over it, about 1 in. deep. The doors are then partly closed with plates of sheet iron, and a light blast is admitted through the ash pit. The coal is quickly ignited by the heat retained in the brick walls of the furnace from the previous charge. When it is burning in all parts a mixture of crushed ore with about half its weight of anthracite is thrown in and spread in a layer 5 to 7 in. thick. The charge of ore and coal is always moistened slightly before throwing into the furnace. The doors at each end are then closed and the blast is gradually increased to the maximum required. About six hours are needed to exhaust the charge. One man attends to a block of six double furnaces and charges them one after the other at intervals of one hour. In this manner the heat of the conduits and other parts of the apparatus is kept very uniform. The ashes are removed

from below the grate once in twenty-four hours. The ore charge for a single furnace is about 250 lb.; for a double furnace about 500 lb.

Collection of the Fume.—The fume escaping through the flue in the top of the arch passes into pipes extending longitudinally over the series of furnaces, which converge into one common pipe, leading to a brick tower where the heavy particles of ash carried along by the blast settle. A further quantity of ash is separated in a second tower. The purified fume is then blown by a fan into a large room, where it cools somewhat, with of course a further deposition of suspended material; from the floor of this room some marketable oxide is collected. The cooled fume goes then to the bag rooms. The general arrangement of the furnaces, towers, cooling-room and bag-room at the Lehigh works at South Bethlehem, Penn., is shown in the accompanying engravings. The second tower is 70 ft. high and about 25 ft. in diameter at the bottom.

The temperature of the collecting flues is so high and the excess of air blown into the furnace is so great that any particles of coal blown into the flues with the gases are burned instantly and no trouble is experienced from their effect on the oxide. Formerly the furnaces were designed so that the gases might be discharged either into the oxide flue or into a chimney at will, and the practice obtained of only turning the gases into the oxide flue when the zinc flame made its appearance, but experience showed that there was no danger in discharging into the oxide flue from the beginning and this is now done commonly. A good deal of care is required on the part of the men operating the furnaces, especially with respect to regulation of the air supply and closing blow holes in the charge so that the air must pass through as uniformly as possible.

The bag rooms are designed with a series of iron pipes, running horizontally under the roof, with outlet tees at regular intervals from which muslin bags are hung. The latter are 30 to 40 ft. long and 2 ft. in diameter. It is reckoned that 200 sq. ft. of bag surface is required for 1 sq. ft. of grate. The bottom of the bag is tied up by a cord, or simply by twisting, so that it may be easily opened for removal of the oxide, which is done by shaking night and morning. After collection from the bags the oxide is sometimes dried in kilns to expel the slight amount of moisture which it absorbs from the furnace gases, and is then bolted to remove fibers of cloth and any other foreign matter that may have been derived from the pipes and bags. It is finally packed for shipment in barrels holding 300 lb.

At the Palmerton works, where there are 24 blocks of oxide furnaces (12 furnaces per block), there are four bag rooms, which have a combined floor area of nearly two acres and contain over 27 miles of bags, the latter

having a total surface of about 850,000 sq. ft. The gas laden with zinc oxide fume is sucked from the furnaces through the settling towers by means of eight exhaust fans, each driven by its own motor. The bag rooms are arranged on three sides of a rectangle, the fourth side of which is occupied by the packing-room. The latter is 62 ft. wide and 143 ft. long, four stories high. The first and fourth floors are employed for the storage of unpacked oxide, which is lifted by means of an electric elevator from the ground floor to the top of the building. Thence it is fed through chutes to bolting machines on the third floor, and from them goes to packing machines on the second floor. The barreled oxide is delivered on platforms at the end of the building, whence it is taken to the storehouses, which have a floor area of 63,000 sq. ft. and a capacity for 9,000,000 lb. of oxide. Bags are sewed and cleaned on the third floor of the packing house. Connected with the latter is a barrel factory, 82×160 ft.

Difficulties in the Process.—While the Wetherill oxide process is apparently so simple it is not free from difficulties. Slight and frequently obscure irregularities may affect the color of the product. Changes in the character of the ore may necessitate different conditions of heat and air supply, which have to be studied carefully. All the volatile constituents of the ore pass into the oxide, which is found to contain traces of sulphur, arsenic, antimony, cadmium and weighable quantities of lead, iron, manganese, lime and magnesia. These may or may not affect the color of the product, which is also dependent upon the physical conditions under which it was volatilized, oxidized and condensed as well as upon its chemical composition. A certain definite relation between the volume of the gas, the quantity of zinc oxide it contains, and the dimension of the flues must be preserved. The heat is controlled by varying the proportion of coal in the charge, but this must also be adjusted to the time the charge is to remain in the furnace. The percentage of oxide in the gas is regulated by the admission of air. The total cross section of flues and capacity of the blast producing machinery must of course be governed always by the maximum grade of ore that is to be treated. In working some ores it is necessary to run as hot as possible without danger of igniting the bags and other parts of the apparatus, while with other ores better results are obtained by running comparatively cold.

Technical Results.—Formerly the average charge used to consist of 100 parts of franklinite, 55·68 parts of reducing coal and 45·95 parts of heating coal. The ore, assaying about 34% ZnO, yielded approximately 24·50% of first quality zinc white, assaying 99·87% ZnO, and 1·5% of second quality, assaying 99·34% ZnO. The residuum amounted to 66·22% of the ore.

Its composition, according to C. F. McKenna,[1] was about 36·43% Fe_2O_3 (25·51% Fe), 15·83% MnO (12·26% Mn) and 9·85% ZnO (7·91% Zn); but the percentage of ZnO was sometimes as high as 11·85 (9·52% Zn). Besides the iron, manganese and zinc oxides, the residuum contains lime, magnesia and alumina.

Since the introduction of the Wetherill process of magnetic separation the franklinite treated has averaged only 28% ZnO and it is expected that better practice will enable 26% ore to be worked. It has been found that about the same yield can be obtained from a works of a given size with 28% ore free from zinc silicate, or nearly so, as can be obtained with 34% ore with zinc silicate present, because no limestone is required in order to make the silicate give up its zinc and consequently the same furnace and fuel work a larger volume of ore. With 28% ore the residuum assays 4·11% Zn, 36·99% Fe, 15·67% Mn and 16·23% SiO_2. In regular practice about 83% of the zinc in the ore is volatilized and recovered as oxide.

Smelting the Residuum for Spiegeleisen.—The residuum is smelted for spiegeleisen in low blast furnaces, it being impossible to use high furnaces, since it is necessary to discharge the gases at a sufficiently high temperature to prevent the zinc oxide from depositing in the throat. In smelting this residuum the zinc is reduced and volatilized in the lower part of the furnace and partly reoxidized higher in the shaft is carried off with the escaping gas. It is only partly reoxidized, large quantities being carried over as blue powder containing sometimes as much as 90% Zn. The downcomer of the furnace leads the zinc-laden gases into condensers which are installed in duplicate so that one set may be cleaned while the other is in use. At the zinc oxide works at Newark, N. J., each set consists of eight clusters of nine vertical pipes, connected at top and bottom with separate rectangular cast iron chambers. There are thus 72 pipes and 16 boxes in each set. The pipes are made of sheet steel, 12 in. in diameter and 23 ft. long. The lower boxes are 4 ft. 10 in. square and 7 ft. 3 in. high. About one third of their interior is filled with brick work sloping to the discharge door. The latter is 2×3 ft., hung at the top and opening outward. In the tops of the upper boxes there are nine circular holes, 11 in. in diameter, closed by lids weighing 50 lb., which are free to be blown off in case of explosion of gas in the condenser. From the last box of the set a 36 in. gas main leads to the hot blast stoves and boilers. The condensers yield chiefly a yellowish-brown dust, containing about 75% ZnO, which in quantity amounts to about 2·3% of the weight of the ore charged into the Wetherill furnaces.

The spiegeleisen blast furnaces of the older works are 42 ft. high, 7 ft. in

[1] School of Mines Quarterly, XVII, p. 171.

diameter at the throat, 8 ft. at the bosh, and 6 ft. at the tuyeres. The blast, which is preheated in a pipe-stove to about 480° C., is blown in through five 4 in. tuyeres. The top of the furnace is closed by an ordinary cup and cone charger. The bosh, which is constructed of brick, 9 in. thick, is protected by an outer water jacket extending 3 ft. above the tuyeres. In smelting the residuum from 100 tons of franklinite, amounting to 66·22 tons, there is obtained 31·72 tons of spiegeleisen, 57·8 tons of slag, and 2·32 tons of flue dust assaying 74·16% ZnO; this oxide is too impure to be used for paint, and is therefore sent to the spelter furnaces. In 24 hours there is obtained 10 tons (22,400 lb.) of spiegeleisen per furnace, for the production of which there are required 20·86 tons of residuum, 11·47 tons of limestone, and 20·86 tons of anthracite. The furnace gas is burned under the steam boilers. The plans for the new Palmerton works contemplate the erection of four blast furnaces, each 60 ft. high and 14 ft. in diameter at the bosh, which are estimated to have a capacity of 72,000 long tons of spiegeleisen per annum. The spiegeleisen produced in the older works assays about 14·96% Mn, 0·412% Si and 0·048% P.

E. O. Bartlett proposed to reverse the usual method of making zinc oxide directly from the ore, in which the blast is introduced beneath the grate, by blowing on a charge placed on glowing coal resting on a suitable grate and drawing off the fumes through the grate and ash pit into settling chambers and filtering apparatus.[1]

W. B. Middleton patented a special furnace for the direct production of zinc white from ore. This furnace is of the reverberatory type, but is without a fire bridge and has a fire box about 3 ft. deep. The fire box is filled with fuel, and a charge of roasted blende, or calamine, is put on the hearth. When it becomes hot it is raked off the hearth on to the top of the incandescent fuel, and a fresh charge is put into the furnace, more fuel being thrown into the fire box at the same time. As soon as the hot ore reaches the fuel zinc flames are evolved and travel over the hearth into a suitable condensing chamber. The fire bars of the furnace are carried loosely on notched bearers, beyond which their ends project, so that they can be turned by means of levers in order to break up and remove the clinker.[2]

MANUFACTURE FROM SPELTER.—In the manufacture of zinc white from spelter in Upper Silesia the metal is heated in retorts to its boiling point and the vapor issuing from the retorts is oxidized by air; the zinc oxide thus formed is collected in settling chambers. Since the crude Silesian zinc always contains lead, of which the oxide gives the zinc white a yellowish

[1] United States patent, No. 515,043, Feb. 20, 1894.
[2] British patent, No. 12,274, June 15, 1901.

cast, carbonic oxide gas from burning coke is introduced into the retort, to convert the lead vapor into lead carbonate, which being heavier than the zinc oxide settles first. The zinc oxide being discolored by the particles of soot from the coke smoke must be glowed in order to make a marketable product. In order to avoid the latter process C. Freitag[1] introduced a blast of air into the pipe conveying the coke gas, whereby the latter is completely burned and a quicker oxidation of the zinc takes place. The Antonienhütte, which is the sole producer in Upper Silesia of zinc white suitable for use as a pigment, has 11 furnaces, each comprising 11 muffles. The oxide is collected by filtration through bags as in the American practice. The output is about 1000 metric tons per annum.

Manufacture of Zinc White in Belgium.—The Société Anonyme de la Vieille Montagne produces annually between 6000 and 8000 metric tons of zinc white in a factory connected with its Valentin-Cocq smeltery. This is made entirely by the burning of spelter. The latter is volatilized in a special form of retort; the vapor issuing from the orifice of the latter is oxidized, and the fume collected by a current of air is conveyed through a series of pipes into long settling chambers, where it is deposited. The chambers have hopper bottoms terminating in tubes to which bags, ordinarily tied up at the lower extremity, are attached. The oxide is removed by opening the bags and permitting it to run into a cask or any other suitable receptacle. The purity of the oxide depends a good deal upon the distance it is carried in the settling chamber before deposition, for which purpose intermediate chambers are interposed between the retorts and the final settling chamber so that incompletely oxidized zinc can be collected separately. The completely oxidized product is further refined by levigation, which affords final products of various qualities. The latter range in composition from 99·695 to 99·995% ZnO (80·002 to 80·243% Zn), 0·20 to 0·002% PbO (0·185 to 0·0019% Pb), 0·1 to 0% Cd, and 0·005 to 0·003% Fe_2O_3. The spelter from which it is made ranges from 2 to 0·04% Pb, 0·04 to 0·012% Fe and 0·07 to 0% Cd.

Zinc Sulphate (Zinc Vitriol).

Zinc sulphate is prepared ordinarily by dissolving spelter, or sheet zinc scrap, in sulphuric acid, evaporating the solution and crystallizing out the salt, $ZnSO_4+7H_2O$, which is the ordinary "zinc vitriol" or "white vitriol" of commerce. It is also made from zinky waste products and direct from ore. In making the sulphate from spelter a solution is readily obtained,

[1] German patent, No. 42,564.

from which only the black floculent matter has to be filtered off, a very simple matter; in making it direct from ore the solution has to be purified from iron, which is a more or less troublesome process.

In the United States zinc sulphate is made on rather a large scale at the Argentine works of the American Smelting & Refining Co., where zinc oxide, mixed with a certain quantity of lead oxide, that is burned off in the refining furnaces from the lead which has previously been desilverized by means of zinc in kettles by the usual process, is collected and treated with sulphuric acid. The zinc is taken into solution as sulphate and is separated from the insoluble lead sulphate by filter-pressing. The zinc solution is then conducted into water-cooled tanks, where it is crystallized, the process being conducted in such a way as to produce the small crystals which are required by the trade supplied from these works. This trade is chiefly with manufacturers of glue. The cakes of lead sulphate discharged from the filter presses are returned to the regular lead smelting process. The production of zinc sulphate at Argentine by this process amounted to 145 tons of 2000 lb. in 1898, in which year the business was first undertaken, and 528 tons in 1899.

A similar method of preparing zinc vitriol is in use at the lead smelting works at Lautenthal in the Upper Harz, where the zinc crust resulting from the desilverization of base bullion is converted by steam into rich lead and argentiferous zinc oxide. The latter is treated with dilute sulphuric acid, in lead-lined wooden vats, whereby the silver contents are collected in the residual slime while the zinc is taken into solution. Ferrous oxide in the solution is peroxidized by the addition of fused saltpetre and manganese dioxide, and is precipitated and separated as ferric oxide. The purified solution is then neutralized with zinc dust before crystallization of the zinc sulphate, since that salt when employed for the manufacture of lithophone must not be too acid.

MANUFACTURE DIRECTLY FROM ORE.—In Germany there is a considerable production of zinc sulphate directly from ore at the Herzog Julius works at Astfeld and the Frau Sophien works at Langelsheim, both places being near Goslar in the Lower Harz, where mixed sulphide ore from the Rammelsberg mine, which is not far distant, is treated. Inasmuch as the practice at those works is one of the best examples of the hydrometallurgical extraction of zinc from mixed sulphide ores, the process employed at them will be described in detail as carried out at the time of a visit in 1893. I have been assisted also in preparing this account by Professor Bruno Kerl's paper on the "Production of Zinc Vitriol from the Zinc-Lead Ores of the Harz," in *The Mineral Industry*, Vol. IV.

Character of the Rammelsberg Ore.—The Rammelsberg deposit, near Goslar, occurs in a schist formation of the Lower Devonian, and consists of an intimate mixture of gold and silver bearing pyrite, chalcopyrite, blende and galena, with gangue chiefly of heavy spar and intermixed country rock (schist). Owing to the intimate mixture of the barite and blende with the other sulphides nothing but a simple mechanical preparation of the ore is attempted, this consisting merely of a sorting according to the predominant mineral (copper ore, lead ore, mixed or copper-lead ore, pyritous ore, etc.). Each kind of ore is again sorted, according to size of the pieces, into Stufferz (pieces of 8 kg. weight), Bergkern (pieces the size of the first), Waschkern (egg-size pieces), Graupel (pieces from the size of strawberries to hazel nuts), and Schlieg (or fines). The copper ore is separated according to its copper contents into three kinds, No. 1, No. 2 and No. 3; the lead ore into pure sulphide and that with much heavy spar (Grauerze).

Of the ores above described, the copper, mixed, and pyritous ores (the mixed ores predominating so that the average tenor of all ores does not exceed 5% Cu) are sent for reduction to the Okerhütte, and the lead ores to the Herzog Julius and Frau Sophienhütte. In connection with the metallurgy of zinc only the treatment of the latter is of interest. Some references to the metallurgical processes employed in the Lower Harz are given in the subjoined footnote.[1]

At the Herzog Julius and Frau Sophienhütte is treated the zinc-lead ore (consisting of about 27% blende, 11·5% galena, 1·5% chalcopyrite, 1·4% pyrite and 46% barytes, calcite and clay shale), about 90% of which is Stufferze and Bergkern and the remainder fines, the whole averaging 10% Pb, 18% Zn, 5% Fe, 0·3% Cu, 0·013 to 0·016% Ag and 13 to 16% S. The combined sulphides (galena, blende, chalcopyrite and pyrite) constitute about 50% of the mass of the ore, the other half consisting chiefly of heavy spar with a little calcite in the coarse ore, and chiefly of schist (country rock) in the fine. The Stufferze and Bergkern are richer in blende than the Waschkern, Graupel and Schlieg. Since the sampling of heaps of ore of such different size is uncertain, the ore is taken by the works from the mine at a fixed price per ton.

The scheme for the treatment of this ore is based on smelting for lead by the roast reduction process, but since zinc blende has a disastrous influence

[1] Wimmer, Ueber den Rammelsberg Bergbau, and Bräuning, Ueber die Okerschen Hüttenwerke; Siegemann, Ueber die Herzog Julius und Frau Sophienhütten, 35 pp., Hanover, 1895; Kerl, Die Rammelsberger Hüttenprozesse am Communion-Unterharzer, 2d edition; Bräuning, "Die Unterharzer Hüttenprozesse," in Zts. f. d. Berg-Hütten-u. Salinenwesen im Preuss. Staat., XXV, 13 to 69; Mittheilungen über den Communion Unterharzer Bergbaue und Hüttenbetrieb für den VI Allgemeinen Deutscher Bergmannstag.

upon the separation of the lead and silver and the formation of a good slag, and also interrupts the furnace campaign by the formation of crusts and scaffolds, the plan is adopted to convert the zinc in the roasting so far as possible into sulphate, which can be leached out with water, leaving thereby an ore capable of fairly easy smelting.

Roasting.—In determining the roasting process as a preliminary to the above scheme various considerations, especially the desirability of producing the greatest possible quantity of soluble zinc sulphate, led to the adoption of heap-roasting, notwithstanding its drawbacks of imperfect desulphurization, long duration and development of sulphurous fumes in the open air. This method is still employed both at Astfeld and Langelsheim. The abundant quantity of lump ore for building the heaps and the high sulphur contents of the ore make a long maintenance of heat in the roast heaps possible if the latter are built sufficiently large.

In building a roast heap a foundation of slag and clay is first tamped down, upon which is laid a thin sheet of dry, fine ore; on this layer (30 to 40 cm. high) is placed a rectangular bed of cordwood (40 to 50 cm. high, or 35 to 40 cu. m.) and fagots, the sides of the bed being 10 to 12 m. long, so arranged that the sticks form air channels or flues. Upon this bed are placed, one over the other, 250 tons of Stufferze, 100 tons Bergkern, 14 tons Graupel and 20 tons Schlieg, so that the heap of 400 tons of ore is arranged in the form of a truncated pyramid 2 m. high, the base covering 11·5 m. square and the upper surface 4 m. square, the sloping sides being 4·5 m. long. After covering the sides with a thick layer of sifted ore from a previous roasting, the wood is kindled from the windward side. This must burn up in 24 to 36 hours in order that the sheet of ore lying above it may be ignited uniformly, after which the oxidation of the sulphur maintains the requisite roasting temperature.

Since with the proportionately large heap, and the size of the pieces of ore which compose it, the air cannot penetrate everywhere, a part of the sulphur of the pyrites is merely volatilized and condenses near the cool upper surface, beginning to show about three or four weeks after the burning of the wood. As soon as the layer of Schlieg is saturated to a depth of about 20 cm., 30 hemispherical colanders, 30 cm. in diameter and 20 cm. deep, are tamped in the upper surface in rows, and in these up to about the end of the third month molten sulphur collects. This is from time to time ladled into moistened wooden vessels and carried to the sulphur refinery common to the two works. The sulphurous acid fumes escaping from the heap also cease after about three months, but are still developed in the interior and are consumed in the formation of sulphates. Eight or nine

months after the burning of the wood the cooling of the heap and entire cessation of fumes indicate the end of the roasting, and the heap is then broken down by means of picks and the loosened material is raked over to separate the fines, which form the leaching product, from the coarse pieces. The fines are due especially to the rich pyritous ore, which is decomposed in consequence of the more abundant formation of sulphates.

The still incompletely roasted lumps are broken up with long hammers and subjected to a second roasting on a bed of wood in heaps of about 10 m. width, 15 to 25 m. length and 3 m. height, consisting of 500 tons and more. These heaps are built under sheds and no covering of fine roasted ore is put upon them. They are so built in order that they may not be cooled down by rains and atmospheric exposure, the low sulphur content of the material furnishing but little heat, and also to guard against leaching out of zinc vitriol. The product of the second roasting is handled in the same manner as that of the first, and the coarse material sorted out is sledged and roasted a third time. The second roasting lasts six to eight weeks, the third three to six weeks. After each roasting the fines are separated, until at last the lumps still remaining contain less than 5% S; the thrice-roasted ore is then sent to the smelting furnaces. The fines from the second and third roasting are poorer in sulphates (containing 15 and 12% respectively) than the fines from the first (which contain 20 to 25%), being richer in heavy spar, of which the lumps do not crumble in the first roasting, and also in oxides, which are formed from the sulphate in the second and third heaps in the absence of a covering of fine ore. Richest of all in sulphates is the covering of the first heap, which contains from 50 to 60%. In the finest almost half of the zinc is converted into a sulphate soluble in water, existing together with iron sulphate, and it follows that the solutions derived from the last roastings are purer than those from the first. In roasting 100 tons of raw ore an average of 13·9 cu. m. of brushwood (fagots) and 212 pieces of cordwood are required.

Leaching Roasted Ore.—The leaching of the roasted ore was formerly carried out in two rows of vats, 5·50×1·80×0·53 m., arranged in terrace form and stirred by hand with wooden paddles, but in 1888, in order to quicken and cheapen the process, lead-lined wrought iron barrels, 1 m. in diameter and 1·4 m. long, revolved mechanically, were introduced. These barrels have horizontal ribs on their inner sides, which serve to strengthen them and also to agitate the pulp. The barrel is filled from a hopper holding the amount of the charge (1200 kg.) provided with a gate at the bottom through which the pulp runs directly into the opening of the barrel. The barrel is filled with warm weak solution and closed, after which it is

revolved for 20 minutes, 20 times per minute, by gearing driven from the main line shaft. After the solution is completed the barrel is thrown out of gear and turned by a crank until the opening comes uppermost. The pulp is then allowed to stand until the coarsest has settled, when the liquor, 35 to 40° B. in strength, is drawn off through a tap and carried through a movable launder into a masonry conduit. The barrel is again filled three quarters full with warm weak solution and rotated, the second solution, 20 to 25° B. in strength, being drawn off in the same manner as the first, with which it is mixed so that the whole has a strength of about 30° B. and is free from danger of crystallization of the salts in solution. The liquor, which clears itself somewhat in the masonry conduits, is drawn into a deep settling basin. After the barrel residue has been treated a third and fourth time with warm water and the solutions (respectively 10 to 15° B. and 7 to 10° B. strong) have been drawn off into the weak solution basin, whence the liquor is pumped for treating fresh ore, the exhausted ore is discharged into a car beneath the barrel.

Evaporation and Crystallization.—For the separation of the iron from the crude solution,[1] the latter is heated to 80° to 90° C. for 20 or 24 hours in lead pans, supported on iron plates 4 m. long, 3 m. wide and 0·62 m. deep (7·2 cu. m. capacity), whereby basic ferric sulphate, together with sulphate of lime and other insoluble salts, are separated. The liquor is then drawn off through a tap in the lower part of the pan, or by means of a siphon, into lead-lined wooden vats of 7 cu. m. capacity to settle the precipitated salts, the clear liquor being tapped out into a basin, whence it is pumped into lead-lined iron pans of the same size as those in which the insoluble salts are separated, where it undergoes concentration. The vaporized water is replaced by solution of 30° B. until at the end of about 36 hours a concentration of 47 to 50° B. is attained. The hot concentrated liquor is then run into lead-lined wooden vats which have V shape bottoms. These vats are 1·75 m. wide, 7 m. long, 0·61 m. deep in the middle and 0·43 m. at the sides, having a capacity of 6·62 cu. m. After six or seven days the zinc vitriol has crystallized out in rhombic prisms, when the mother liquor is tapped out through a hole in the bottom into a reservoir, whence it is returned to the concentrating pans until its tenor in foreign salts (iron and copper sulphates) becomes too high. The zinc-vitriol crystals are shoveled into cars and carried to the drying-room, where they are emptied upon a sloping platform, whence the mother liquid drains away. The vitriol is

[1] An attempt has been made to separate the iron by means of bleaching powder. This, however, proved too costly, and besides formed zinc chloride, which does not crystallize out and is lost in the mother liquor. Otherwise bleach is an effective oxidizing agent.

composed with slight variation as follows: ZnO, 27%; MnO, 0·50%; FeO, 0·05%; SO_3, 27·95%; H_2O, 44%; total, 99·50%. The 0·50% unaccounted for consists among other things of about 0·05% cadmium oxide and 0·01% cobalt and nickel protoxide. It is shipped in sacks or in bulk. When chemically pure, crystallized zinc sulphate contains 28·24% ZnO, 27·87% SO_3 and 43·89% H_2O.

Preparation of Calcined Vitriol.—In order to save transportation on the water of crystallization, a portion of the latter is sometimes eliminated by calcination. The crystals are melted in their water of crystallization in a copper kettle 1 m. deep and 1·18 m. in diameter, without allowing the solution to come to the boiling point, in which case zinc sulphate would stick to the sides of the kettle. As soon as the charge has become fluid, it is skimmed and transferred by means of a copper ladle into a trough, through which it is conveyed to a vat with sloping sides, where it is stirred with a wooden paddle for two or three hours, or until it sets. By this process about one half of the water of crystallization is driven off.

Smelting the Residuum.—The leached ore discharged from the barrels is collected in sumps, whence it is periodically removed and after being dried is smelted for lead in blast furnaces. It is still high in zinc, involving peculiarities in the smelting process, which fall, however, into the metallurgy of lead rather than into that of zinc.

USES OF ZINC SULPHATE.—Zinc sulphate is used somewhat as a mordant in dyeing and calico printing; as a disinfectant; for preserving and clarifying glue solutions; in medicine as an astringent, and in lotions; in the preparation of dryers for "boiled oils"; and to some extent as a preservative for hides. In Germany the largest use is for the manufacture of lithophone, for which purpose there is also an important consumption in the United States. Zinc sulphate also finds employment in the chemical industry as a precipitant of contaminating sulphides in certain processes.

Crystallized zinc sulphate is marketed in the United States in kegs of 200 lb. and in barrels of 350 lb. The wholesale price at New York in April, 1901, was 2·5@2·75c. per pound. Probably 2@3c. per lb. would be a fair range. Reckoning zinc as costing 4c. per lb. and sulphuric acid of 60° B. as $8 per 2000 lb., the cost of the raw material in a pound of crystallized sulphate is 1·075c.

LITHOPHONE.

A considerable quantity of zinc is employed as a pigment and for other purposes under the name of "lithophone," which is a mixture of zinc sulphide and barium sulphate, prepared by a double chemical precipitation.

A portion of the zinc sulphide is likely to be unavoidably replaced by zinc oxide. Theoretically the precipitate should consist of about 30% ZnS and 70% $BaSO_4$; a commercial product assayed 68% $BaSO_4$, 24·85% ZnS and 7·28% ZnO.[1] Different makes vary somewhat in their composition.[2] Zinc sulphate is employed as the source of the zinc, while the barium sulphide is prepared by heating ground barytes mixed with fine coal and leaching the frit with hot water.

The manufacturers at Schweinfurth in Bavaria and Schoeningen in Brunswick prepare lithophone by precipitating zinc sulphide and barium sulphate from a mixture of solutions of zinc sulphate and barium sulphide the precipitate being washed, dried and calcined. The furnaced product is thrown into water while still hot; it is then ground very finely and dried.

According to another process, that of Cowley,[3] equivalent quantities of solutions of zinc sulphate and barium sulphide are mixed and 0·5 to 1% of freshly prepared magnesia and finely pulverized salt are added to the precipitate before filtration, which is done by means of a filter press; to the dried precipitate 3% of ammonium chloride is added and the whole is then glowed. It is said that color prepared in this manner does not darken on exposure to the sunlight and of course it is unacted upon by gases containing sulphureted hydrogen.

Lithophone is consumed in very large quantities for the manufacture of enamel paints, linoleum and table oilcloth. It is a saturated compound, and it does not continue to oxidize linseed oil; it might therefore be termed a non-siccative. In making enamel paints where rosin varnishes are employed, it does not combine with the rosin, and consequently will not saponify. Because of the fact that zinc sulphide is photogenic, lithophone absorbs light and turns grayish black when exposed to brilliant sunlight, but when placed in a dark room, it loses its gray effect and turns white again.[4] Thus, enameled iron bedsteads, which were gray when exposed to sunlight, after dark would appear white.[5]

In the manufacture of lithophone, a great purity of the raw materials is necessary for the production of a pure white precipitate. If the barytes

[1] The Mineral Industry, VII, 88.

[2] By using a larger proportion of barium sulphide than called for by the formula and precipitating the excess with another zinc salt, such as the chloride, or with sodium sulphate, mixtures containing a higher or lower percentage of zinc sulphide are thrown down. The tendency is to skimp the zinc, which is the more expensive ingredient. References as to the methods of analysis for lithophone may be found in Zts. f. angew. Chem. XV, 174; ibid, XV, 802; Journ. Soc. Chem. Ind., XXI, 427; ibid., XXI, 1145; Bull. Soc. Chimique, XXVII, 829; ibid, XXVII, 943; Chem. Ztg. Repert., XXVI, 229; ibid, XXVI, 297.

[3] Chemical News, 1891, pp. 63 and 68.

[4] This phenomenon takes place in the presence of organic matter.

[5] Maximilian Toch, Journ. Soc. Chem. Ind., Jan. 31, 1902, p. 103.

that is used be ferruginous, or if iron be present in the zinc sulphate, the precipitate will be discolored also by iron. Lithophone will not replace zinc white and white lead as a general pigment, because exposure appears to affect it. Aside from that drawback, lithophone is an admirable pigment, having a great covering power and some other excellent qualities. If a compound of zinc oxide and barium sulphate could be prepared chemically, it would be an ideal pigment, inasmuch as it would have the excellent properties of an artificial mixture of zinc white and blanc fixe (precipitated barium sulphate). Processes for the production of such a compound have been invented, but so far as I am aware have not yet been applied practically.[1] However, lithophone is likely to contain a certain proportion of zinc as oxide, instead of sulphide, because barium sulphide undergoes a partial decomposition, when dissolved in water, according to the equation

$$2BaS+2H_2O=Ba(SH)_2+Ba(OH)_2.$$

The hydrosulphide of barium precipitates zinc as sulphide, while the hydroxide throws down the hydrous oxide of zinc.

Lithophone is made in Germany on a large scale. A good deal used to be imported into the United States, but American manufacturers, who are protected by a duty of 1c. per lb., have now secured control of the domestic market.

A variety of lithophone known in the trade as "zinc anhyride," or "zinc barytes," is prepared by intimately mixing chemically equivalent quantities of chalk (calcium carbonate) or witherite (barium carbonate) and finely pulverized zinc sulphate and heating the mixture in a specially designed furnace to light red heat. The mixture first melts in the crystal water of the zinc sulphate which facilitates a thorough incorporation of the ingre-

[1] A recent invention of Charles B. Jacobs holds out great promise in this direction. He discovered that at the high temperature obtainable with the electric furnace three parts of barium sulphate react with one part of barium sulphide, forming barium oxide and sulphurous anhydride. In practice he mixes in enough coal to form the required proportion of sulphide in the furnace charge. The reactions which take place are represented by the following equations:

$$4BaSO_4 + 4C = 3BaSO_4 + BaS + 4CO$$
$$3BaSO_4 + BaS = 4BaO + 4SO_2$$

The barium oxide is dissolved in hot water and is recovered in a pure condition as barium hydrate [$Ba(OH)_2 + 8\ H_2O$] by crystallization from the solution. The sulphurous anhydride gas is available for sulphuric acid manufacture. The electric furnace is of the continuous type. The process is now in use at Niagara Falls, N. Y., by the United Barium Co. The percentage of the barium sulphate converted is said to be so high (97 to 98%) and the cost of production so comparatively low that barium salts will be available at much lower prices than heretofore. Already, after less than a year's operation, it is said to be necessary to increase the plant to a capacity of 60 tons of hydrate per day. Mr. Jacobs states that the pigment made by precipitation of zinc sulphate has greater covering power and is more durable for outside work than zinc white, can be mixed with other pigments, is almost impervious to moisture and is non-porous when mixed with linseed oil.

dients. Continuing the heating the mixture eventually decomposes to calcium or barium sulphate and zinc oxide, carbonic acid being liberated. The product is said to have excellent covering qualities as a paint and has found a ready market.

Sulphophone is a mixture of calcium sulphate and zinc sulphate or oxide, which is prepared in a manner analogous to that employed for lithophone, a solution of calcium sulphide being used instead of barium sulphide.

Zinc Chloride.

Zinc chloride is usually prepared by dissolving scrap zinc in muriatic acid in stone or earthenware vessels. The solution is carefully neutralized and after being warmed to 40 or 50° C. bleaching powder is added to precipitate iron and manganese. The clear solution is then siphoned off and boiled down until all the water has been eliminated and the chloride remains in a molten condition, which requires a temperature slightly in excess of 262° C.—the melting point of $ZnCl_2$. This operation is analogous to the boiling down of caustic soda solutions and production of fused caustic, but although zinc chloride behaves in somewhat the same manner, first frothing up and then settling down to a quiet fusion, there are special difficulties in the process.

If traces of iron in the zinc chloride are unobjectionable the boiling down may be done in iron pots, which are unattacked by the chloride so long as no free acid is present. The latter is likely to be formed, however, inasmuch as the last of the water cannot ordinarily be eliminated without decomposing some of the chloride with the formation of oxychloride and free acid. This not only detracts from the value of the product by the presence of oxychloride and iron, but also causes a loss of chlorhydric acid. Enameled iron pots prevent contamination by iron, but they are expensive and do not last long. The production of pure and perfectly anhydrous chloride is probably best effected by effecting the evaporation in a vacuum.

Zinc Chloride and Potassium Chlorate.—An interesting method for the preparation of zinc chloride in connection with potassium chlorate has been described by K. J. Bayer.[1] In carrying out this process chlorine gas is passed into zinc oxide suspended in cold water, whereby zinc chloride and zinc hypochlorite are first formed, the whole of the zinc going finally into solution according to the following equation:

$$2ZnO+4Cl=ZnCl_2+ZnCl_2O_2.$$

[1] Chem. Ztg., 1895, XIX, 1453 to 1455; Journ. Soc. Chem. Ind., Jan. 31, 1896, p. 32.

The zinc hypochlorite is converted into zinc chloride by potassium chloride, potassium chlorate being formed in the reaction as follows:

$$3ZnCl_2O_2+2KCl=3ZnCl_2+2KClO_3.$$

The potassium chloride may be added to the zinc oxide in the beginning or after the solution of the zinc has been effected, the temperature being maintained at 90 to 95° C. In either case a solution of about 30° B. is obtained which on cooling yields fairly pure crystals of potassium chlorate. On concentrating the mother liquor to about 60° B. the greater part of the remaining potassium chlorate crystallizes out. The mother liquor, then containing only very little potassium chlorate, is treated with chlorhydric acid and evaporated to dryness to obtain zinc chloride. The apparatus required differs but little from that which is used in the manufacture of potassium chlorate by the lime process, the two processes being indeed identical in their reactions with the only difference that zinc oxide is substituted in one for calcium oxide in the other. The evaporation of the zinc chloride solution may be carried out in iron vessels, preferably in a vacuum, if traces of iron are not detrimental. The zinc process is claimed to give a better yield of potassium chlorate than when either lime or magnesia is employed, and of course the whole of the chlorine used is obtained as a marketable product. The zinc chloride which is produced is very pure but may contain a little potassium chloride, which, however, is not detrimental for many purposes.

Uses of Zinc Chloride.—Zinc chloride is employed to some extent for weighting cotton goods. A solution of sp. gr. 1·7, boiled with excess of oxide, forms an oxychloride ($ZnCl_2$,$3ZnO$,$4H_2O$) which dissolves silk, and may be used for separating that fiber from wool. Zinc oxychlorides are also used as pigments, cements and for making artificial teeth. Solutions of zinc chloride were also employed by Mercer for treating cotton in the operation of "mercerizing." A solution of zinc ammonium chloride is used in soldering for the purpose of removing oxide from the surface of the metals. Anhydrous zinc chloride is sometimes employed instead of ammonium chloride as flux in galvanizing. The chief use for zinc chloride is in the preservation of timber by the process known as burnettizing, which was first patented by Sir W. Burnett in 1838.

O. Chanute, the leading American authority on this subject, concluded from his experience that if wooden railway sleepers be injected with reasonable uniformity and the equivalent of 0·5 lb. of dry zinc chloride per cubic foot, as is done in Germany, burnettizing makes them last 10 to 12 years

in the track, with ordinary exposure, while perhaps half that quantity will produce the same result in the more arid regions of the United States. The zinc-tannin process will impart to them a life of 12 to 14 years and the zinc-creosote process may extend it to 14 to 16 years.[1]

Granulated zinc chloride is commonly packed in barrels containing 700 lb. or in cases containing 220 lb. Its value at New York in April, 1901, was 5@5¼c. per lb., according to the size of the order. The chloride to be used for the preservation of wood must be free from iron and purchasers generally stipulate as to the percentage of oxychloride.

[1] Eng. and Min. Journ., 1900, LXX, 606.

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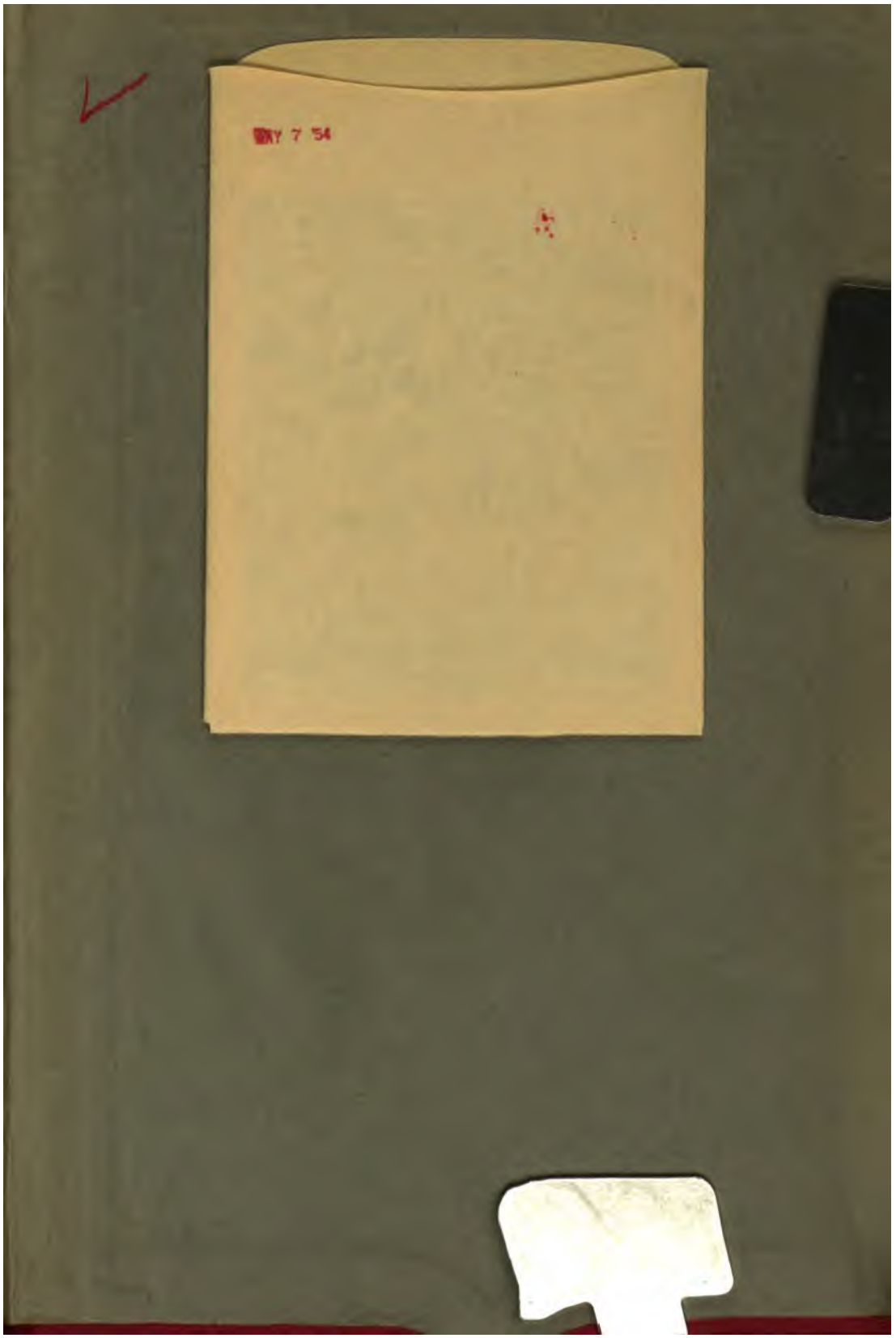
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